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(NONMETALLICS)

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FOREWORD

Indicative of the growth of interest among mining engineers in the mining and treatment of the industrial minerals—nonmetallics—this volume is the first devoted exclusively to those commodities to be issued by the Institute. It is the earnest hope of the members of the Industrial Minerals Division that this volume will initiate a valuable reference library for the operating engineer in this field as well as for instructors and students in our educational institutions.

The volume contains thirty-eight papers. Ten of these relate to mining methods employed on the mineral aggregates and building stone and eight to treatment methods. Following these are fifteen papers arranged as symposiums on phosphates, potash, and salt. In addition, there are six papers dealing with the mining or treatment of other industrial minerals.

Also, for the first time in a TRANSACTIONS volume, the TECHNICAL PUBLICATION (T.P.) or CONTRIBUTION numbers of the papers as originally issued are given in the table of contents and at the foot of the title page of the respective papers.

Papers dealing with industrial minerals were last published in Volume 109 (1934); those in the present volume have been selected from the pamphlet publications appearing in the interval.

Credit for this accumulation of material is due to the chairmen and members of the technical committees of the Industrial Minerals Division during the years represented.

To H. I. Smith, Chairman of the Volume Committee, is due credit for careful and painstaking selection and editing of both manuscripts and comments and their arrangement in the volume.

The Institute and the profession are indebted to the forty-nine authors who contributed these papers for an excellent beginning of an eventual comprehensive library of technical information concerning industrial minerals.

J. R. THOENEN, *Chairman,*
Industrial Minerals Division.

COLLEGE PARK, MD.
July 1, 1938

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The Functions of Power Scrapers and Slackline Cableway Excavators

By HARRY A. ROE*

(New York Meeting, February, 1937)

THE power drag scrapers and the slackline cableway excavator have been called "long-range excavators." Broadly, their field of usefulness is restricted to work in which their long range of action permits them to operate economically, and which other, possibly more conventional, excavating units cannot do. Both of these machines dig or lift material and transport it to a remote dumping point. They combine these two functions in a single machine, nearly always in a single straight-line motion.

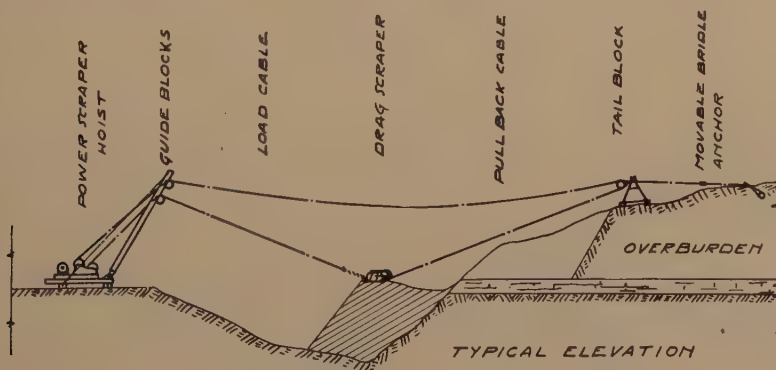


FIG. 1.—SIMPLE FORM OF POWER DRAG SCRAPER.

The purpose of this paper is to examine the important features that distinguish the machines in this group from each other, as well as from other groups.

THE MACHINES

The "power drag scraper" is a long-range excavating machine that employs a bottomless drag scraper operated back and forth between digging and dumping points by two haulage cables. The typical arrangement is that of Fig. 1. This is the "slusher" of the underground mine, as well as the "dragline scraper" of the sand and gravel industry, and other open-pit mining operations.

Manuscript received at the office of the Institute Nov. 23, 1936. Issued as T.P. 799 in MINING TECHNOLOGY, May, 1937.

* Chief Engineer, Sauerman Bros., Inc., Chicago, Ill.

The "gravity-return scraper" is a long-range excavating machine that employs a bottomless drag scraper, hauled to the dump by a cable, but returned from the dump to the digging point by gravity while suspended from a carrier on a track cable or slackline. The typical arrangement is that of Fig. 2. This gravity-return scraper is by no means a mongrel machine, although obviously it results from cross-breeding the power drag scraper and the slackline cableway excavator.

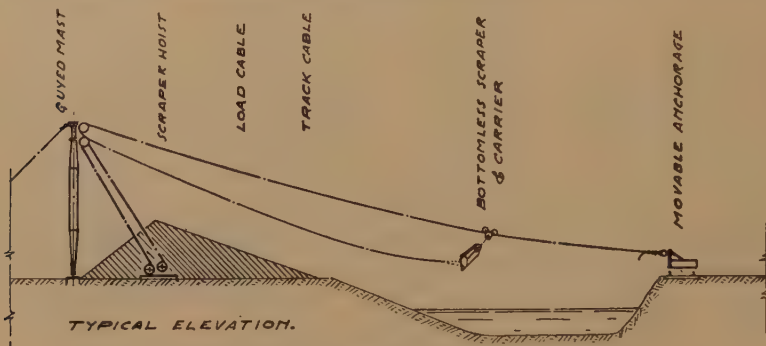


FIG. 2.—GRAVITY-RETURN SCRAPER—RADIAL.

The "slackline cableway excavator" is a long-range excavating machine in which the loaded bucket is conveyed to the dumping point while suspended in the air from a carrier running on a track cable or slackline. The typical arrangement is that of Fig. 3. This machine employs a dragline bucket having a bottom. The bucket is very similar to the dragline buckets used on revolving draglines, except in the method



FIG. 3.—TYPICAL SLACKLINE CABLEWAY EXCAVATOR—RADIAL.

used in dumping the load. The cableway bucket is suspended from the carrier so that it will dump at any point along the track cable when it comes in contact with a stop-button clamped to the cable. Most slackline cableway excavators dump at the upper end of the track cable, but the carriers can be arranged to dump at the lower, or outboard end. The bucket is returned down the track cable by gravity.

The three machine units shown in Figs. 1, 2 and 3 are rather elemental forms. Each operates along radial lines diverging from a relatively fixed

mast, or A-frame, and dumps the excavated material onto a pile or into a processing plant or haulage system near the mast. In fact, the usual installation is on just such radial lines. It should be noted, however, that the applications likely to be of the most interest to engineers may



FIG. 4.—DRAG SCRAPER TOWER MACHINE.

not lend themselves to radial operation of the excavating unit. For such applications the three radial long-range excavators of Figs. 1, 2 and 3 take the general forms shown in Figs. 4, 5 and 6, respectively. Self-propelled movable towers at opposite ends of the span are typical,

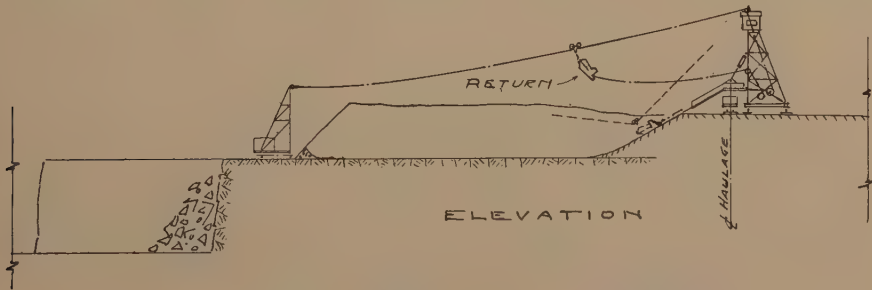


FIG. 5.—GRAVITY-RETURN SCRAPER TOWER MACHINE.

mounted on crawlers when circumstances warrant. Each head tower of Figs. 4, 5 and 6 provides for a different kind of dump, but these variations are interchangeable between the three machines. Such a long-range excavator, designed for more or less continuous movement, is called,

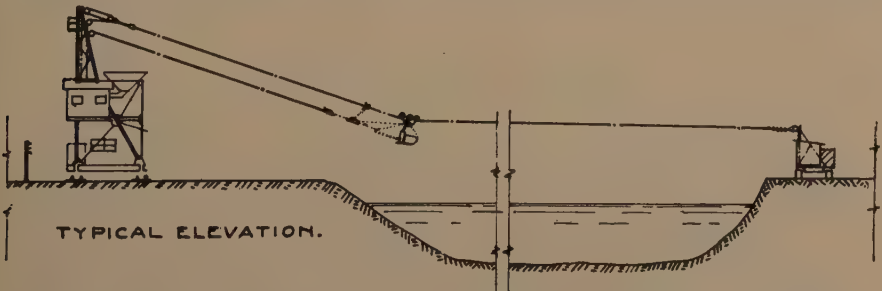


FIG. 6.—SLACKLINE CABLEWAY TOWER MACHINE.

generally, a tower machine, and specifically: drag scraper tower machine (Fig. 4) slackline scraper tower machine (Fig. 5) and slackline cableway tower machine (Fig. 6).

The power drag scraper, the gravity-return scraper, and the slackline cableway excavator do not represent simply three similar forms of the

same machine, nor do they constitute three parallel ways of doing the same work. They are complementary machines in about the same sense that the revolving dragline, the crawler crane and the power shovel are complementary machines. Their fields of application diverge along rather clearly distinguished lines.

APPLICABILITY OF THE DIFFERENT MACHINES

The gravity-return machines are not used underground, because of the limitation of headroom. The simplicity of the drag scraper, coupled with the development of extremely compact hoists, has led to extensive use of the drag scraper in many underground loading operations. The charts of capacity and power consumption (Figs. 9 to 14) apply to well designed slusher operations.

Above ground the drag scraper is preferred to the slackline gravity-return machines for all typical "bank excavations"; that is, work in which the dumping point of the bucket is definitely lower than the digging point. The most frequent application of this kind is that of a sand and gravel plant at the base of a hill deposit (Fig. 7).

There are two popular fallacies with regard to drag-scraper operations. The first of these is related to bank excavation. Dragging loose material down a slope is easy, of course, for a drag scraper. The fallacy comes in assuming that reversing the operation places a drag scraper at a mechanical disadvantage with respect to other methods. There is a substantial difference in the power required for the two operations. Analysis or experiment will show this difference to be almost wholly "useful work," which is unavoidable and quite independent of the method.

A second, and somewhat less frequent, misconception of the drag scraper has to do with working under water. The idea that the bottomless drag scraper is not a suitable tool for excavating materials under water is an error. It is true that underwater excavation incurs certain difficulties not met in dry excavation. As a matter of fact, however, the drag scraper had its first substantial development on the west coast in the excavation of gravel from shallow streams. Handling capacities and unit costs must be discounted somewhat for underwater as against dry operation, but some of the most successful installations from the standpoint of relative economy have been from underwater.

Therefore, the work especially suited to the power drag scraper in the mining of industrial minerals may be summarized as follows: (1) underground slushing, (2) bank excavation, (3) wet or dry excavation over distances that are relatively unfavorable to the longer range slackline gravity-return machines and to the shorter range revolving excavators.

The gravity-return scraper (Fig. 2) shows lower maintenance costs and slightly increased handling capacities, size for size, on the longer hauls in comparison with the drag scraper. Therefore it should have

preference for intermediate and longer spans—say, above 300 ft.—where conditions are otherwise favorable. In underground operations and bank excavations, the conditions are manifestly unfavorable to the gravity-

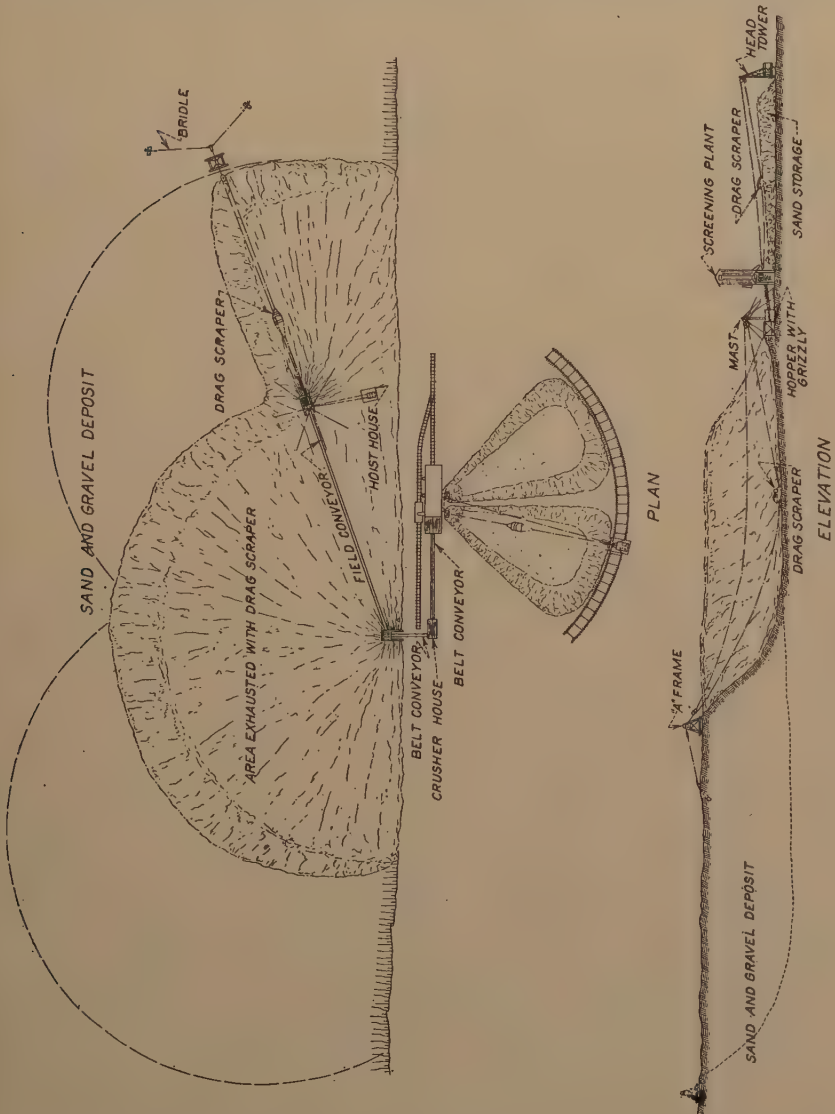


Fig. 7.—EXCAVATING HILL DEPOSIT FOR SAND AND GRAVEL PLANT AT BASE OF HILL.

return scraper. The essential field for this scraper may therefore be summarized as: wet or dry excavation over distances that are relatively unfavorable to the drag scraper, and in which the ability of the slackline cableway to discharge its load into a substantially elevated structure is not the deciding factor.

The slackline cableway excavator is so called because it becomes a conveying cableway as soon as the bucket is properly loaded. The high mechanical efficiency of this conveying portion of the operating cycle results in the highest relative economy on the longer spans. It is infrequently used on spans shorter than 300 ft. Spans of 1000 ft. and over are not uncommon.

The cableway excavator dumps its load most advantageously at the upper end of the inclined track cable; that is to say, at a considerable elevation above the base of the machine. Therefore it has an inherent advantage where delivery of the material to the top of a plant is desirable. This machine digs, conveys, elevates and dumps in one continuous, forward movement. Therefore, its essential field may be said to be: (1) wet or dry excavation on spans upward of 300 ft. where the material is to be taken from relatively deep deposits and delivered to relatively elevated dumping points; (2) other operations in which the elevating and conveying features of the cableway are the deciding factor over the power scraper; (3) stripping or mining operations requiring a dump at the low end of the span and the suspension of the load between the digging and dumping points to clear intervening obstructions.

OPERATING PROBLEMS

During the past 17 years I have seen power scrapers handling a good many different kinds of materials. As would be expected, a reasonable proportion of these power scrapers were not doing their jobs satisfactorily. Why? In the majority of instances the user, the buyer, or the seller had disregarded the most simple axiom in drag-scraper practice; that is, "When the material does not come to the bucket, the bucket must go to the material." In other words, the average drag-scraper installation is sold by the manufacturer on the assumption—often erroneous—that the scraper is going to handle reasonably free-caving material. Provision made in the average installation for controlling and shifting the digging point of the bucket is entirely adequate for caving materials; it is all too frequently inadequate for noncaving, hard, or sloppy materials.

Adequate provision for easy, frequent, and rapid shifting of the line of operation of the bucket, and for the control of the loading point is readily available. In slackline gravity-return machines such provision is most adequately provided by the use of self-propelled tail towers (Figs. 5 and 6). On intermediate and long-span drag-scraper operations (Figs. 1 and 4) movement of the tail block does not always suffice.

A drag scraper does not operate on straight radial or parallel lines, but follows the path of least resistance. The loaded scraper slides sideways and travels in the bottom of the cut. The empty scraper runs back in the old cut, and will leave ridges and a rough and unsightly pit unless skillfully operated. Failure to recognize this tendency and to apply its

corrective is the major source of operating problems. A third operating cable with a flying block (loading block) riding the back-haul cable opposite the digging point, as in Fig. 8, will control the bucket, clean up ridges, and leave a clean pit or stripping job. Control of the flying block by a hand winch has proved quite satisfactory for small installations. Control from the scraper hoist by the operator is faster and more economical for larger installations.

In extremely coarse material, such as blasted rock, and in very hard digging, such as soft shales, there is an added necessity for the ability of the scraper operator to place the bucket at the most advantageous point for loading. It is not only undesirable to dig at the bottom of the cut; it is very much more difficult to do so.

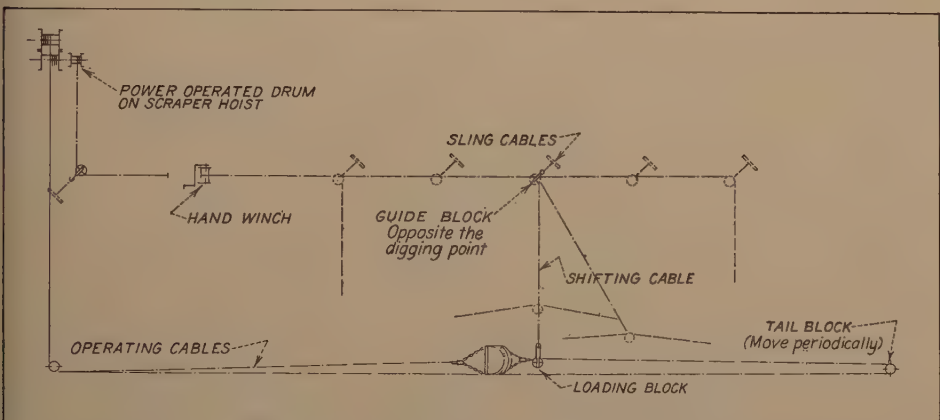


FIG. 8.—SHIFTING DEVICE FOR PREVENTING RIDGES IN DRAG-SCRAPER OPERATION.

Therefore, the feature necessary to practical success most frequently overlooked in power-scraper practice is adequate provision for definite control of the loading point of the scraper.

HANDLING CAPACITIES AND UNIT COSTS

A discussion of handling capacities and unit costs of these long-range excavators offers certain difficulties because of the number of variable factors involved. In addition to the effect of differences in hardness, density, abrasiveness, etc., of the materials handled, there are wide variations in distances, elevations, and time lost for moving. It is not practical to examine these in this kind of paper; indeed, it is scarcely desirable or useful to do so.

The principal factors controlling handling capacity are: (1) size of bucket, (2) speeds of operation, (3) density of material, (4) distance. In a properly balanced machine there is a relationship between factors 1, 2 and 3, which determines the proper size of electric motor, and manufacturers' standards are largely based upon the available sizes of electric motors. For purposes of this paper, therefore, handling capaci-

ties have been reduced to 40° continuous rated motor sizes. In a well designed power scraper or slackline cableway excavator machine the relationship between size of bucket, line speeds and density of material will result in a proper and uniform utilization of these motors.

Fig. 9 shows handling capacities of the power drag scraper in tons (2000 lb.) per hour for *average* hauls up to 600 ft. These capacities vary nearly inversely with the distance, but are increasingly efficient with an increase in distance. Fig. 10 shows similar capacities for the gravity-return scraper on spans up to 800 ft. The shorter time cycle in com-

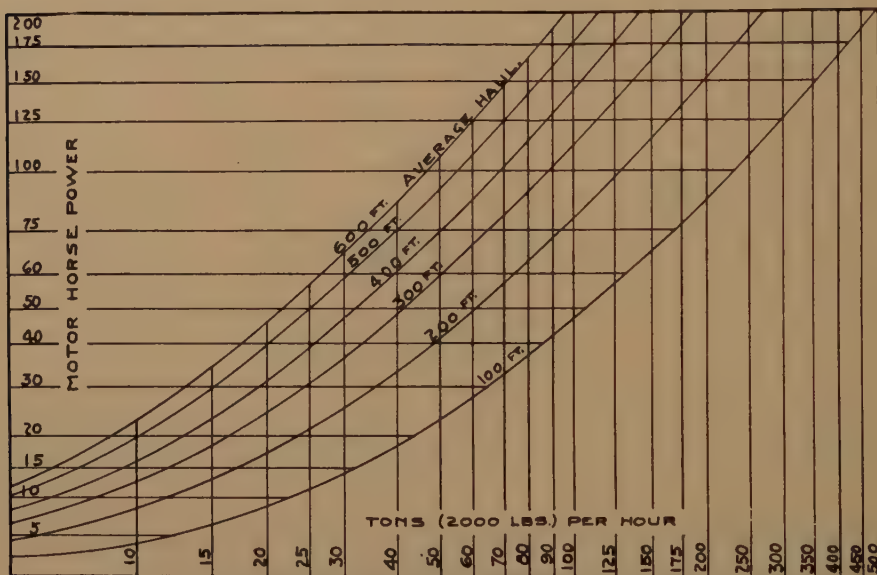


FIG. 9.—HOURLY HANDLING CAPACITIES OF POWER DRAG SCRAPERS.
Tons (2000 lb.) for various motor horsepower.

parison with Fig. 9 is apparent on longer hauls. Fig. 11 shows handling capacities for slackline cableway excavators. This chart is based upon manufacturers' tables and indicates the relative inflexibility of these machines as compared with the power scraper. Fig. 11 is not as accurate as Figs. 9 and 10, because of the variations introduced by changes in span and in the depth and character of digging. All of the capacity charts are reasonably accurate on the basis of hourly capacity in typical situations. For daily outputs, capacities taken from the table should be discounted from nothing in loose-caving dry materials up to about 50 per cent in extremely difficult situations, such as mucking blasted rock.

Figs. 12 and 13 show power consumption for various average distances for drag scrapers and gravity-return scrapers, respectively, and permit a ready comparison of these costs between these two types of scraper machines. In addition, they show the effect on unit power cost of down-grade and up-grade power-scraper operation.

Fig. 14 shows power consumption for various average hauls for slackline cableway excavators where the depth of digging is 10 per cent of the total span of the cableway and the average haul is from 50 to 75 per cent of the span. Under other conditions the power consumption is modified one way or another by the effect upon the time cycle of tensioning the track cable.

The charts for unit power consumption are substantially correct for a wide range of situations and materials. They were worked up from data obtained from a similar range of installations. For example, on an actual installation of a 3-cu. yd. drag scraper handling ore weighing from 125 to 150 lb. per cubic foot, the unit power consumption varied little

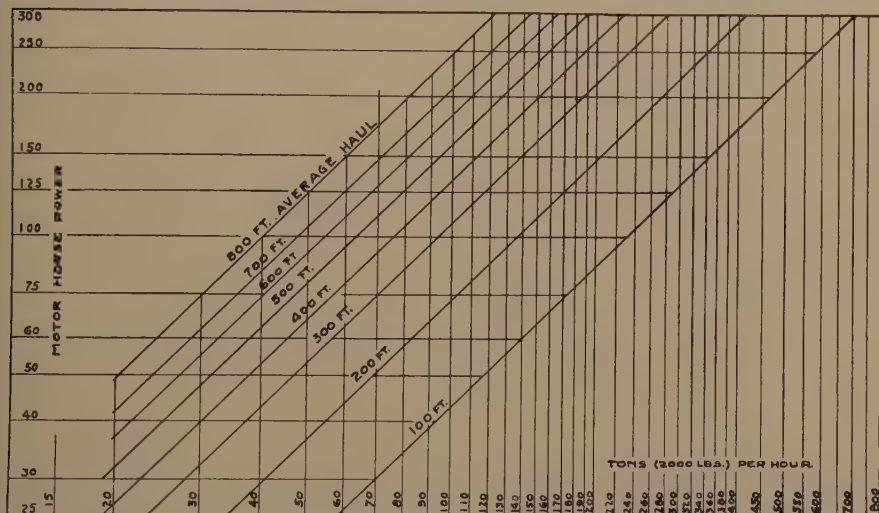


FIG. 10.—HANDLING CAPACITIES FOR GRAVITY-RETURN SCRAPERS.
Tons (2000 lb.) for various motor horsepower.

for the change in density, and agreed with Fig. 9 within the variation expected between plants. Incidentally, it was less than the table.

The determination of labor cost must of necessity be considered with regard to a specific situation for any one of these machines, and any comparison of labor requirements of long-range excavators with those of other excavating units must of necessity involve the associated transport problems.

The actual operation of a drag scraper, a gravity-return scraper, or a slackline cableway is a one-man job. Experienced operators are not readily available but competent operators are readily trained. In free-caving materials of considerable depth no other labor is required for considerable intervals of time. Occasional common labor is required to shift the line of operation to avoid exhaustion of the material in one cut.

On tower machines mounted on tracks, a ground crew is required for incidental grading and for laying track. The largest tower machines are served by crews of seven men, consisting of: an operator, a platform man

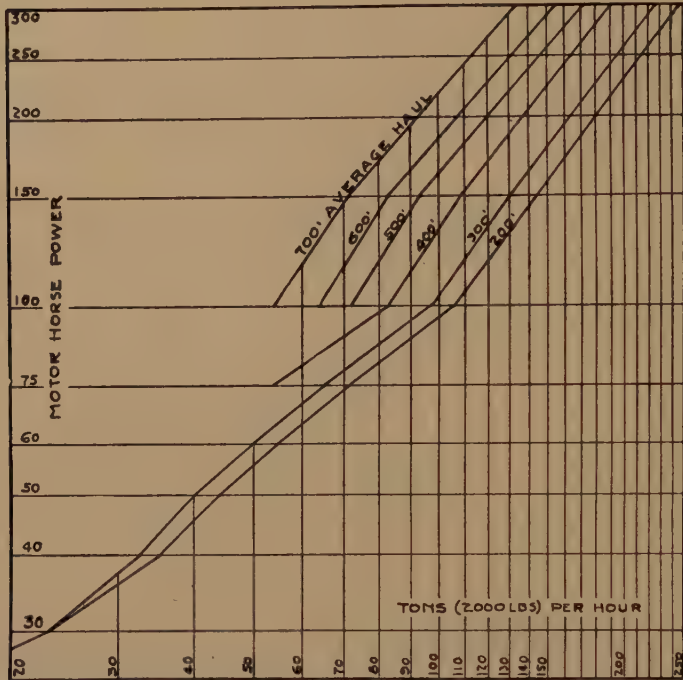


FIG. 11.—HANDLING CAPACITIES OF SLACKLINE CABLEWAY EXCAVATORS, FOR VARIOUS HORSEPOWERS.

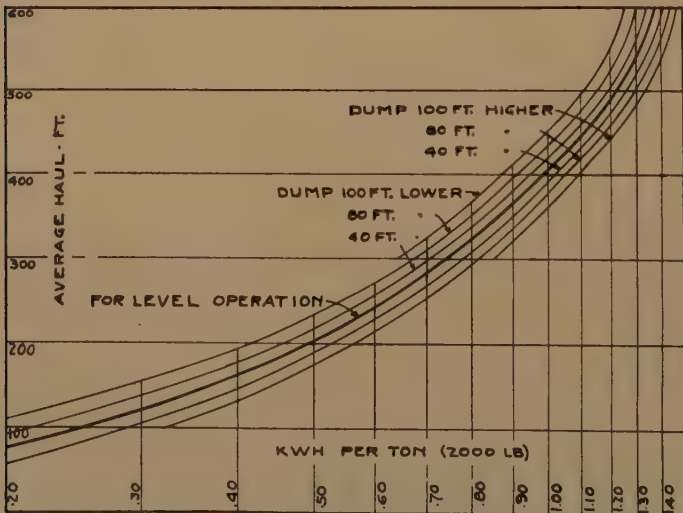


FIG. 12.—POWER CONSUMPTION FOR POWER DRAG SCRAPERS.
Kilowatt-hours per ton (2000 lb.) for various average hauls.

(mechanic), a tail-tower man (skilled to some degree), a ground crew (labor) of four men.

The labor per ton in man-hours is readily arrived at from the capacity chart and an estimate of the men per shift.

The third item in direct operating cost is repair and maintenance. For typical sand and gravel plants this will run from \$0.0075 to \$0.0150 per ton, depending upon the average haul, abrasiveness of the material,

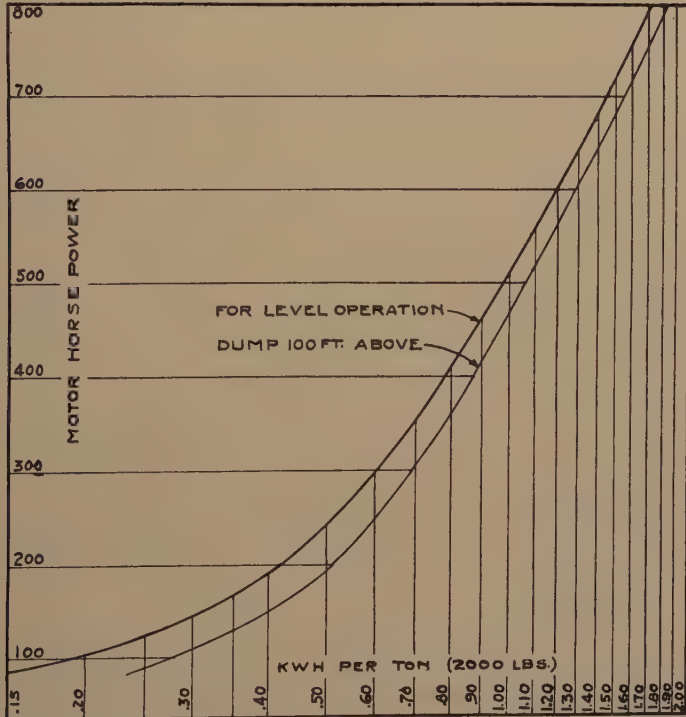


FIG. 13.—POWER CONSUMPTION FOR SLACKLINE CABLEWAY EXCAVATORS.
Kilowatt-hours per ton (2000 lb.) for various average distances.

and the size of the machine. A drag scraper of 1 cu. yd. capacity, producing 10 cars of finished material per day on a 200-ft. average haul, will show a cost of about \$0.0100 per ton over a long period of time for replacement of wearing parts and ordinary maintenance. More abrasive materials will show greater costs over the same haul, and vice versa. The gravity-return scraper will show maintenance costs up to 50 per cent under the drag scraper. The slackline cableway excavator will show maintenance costs of about \$0.010 per ton on an average haul of 350 ft. on the same basis.

I shall not attempt any discussion of plant costs and fixed charges. My usual practice is to regard such expendible items as wire rope and bucket replacement as proper repair and maintenance, and to write off the plant over 15 years, or against a fixed tonnage, as circumstance dictates.

From the engineer's point of view, a study of the economies of the power scraper and the slackline cableway excavator presents a paradox. Estimates of costs, and actual costs secured from successful plants, show clearly that this group of machines is capable of relatively low-cost production. At the same time, the extent of the commercial use and exploitation of these machines does not make a corresponding comparison with combinations of other units which together will do the same work. As engineers we are not concerned with commercial development, as such. It is necessary, however, to inquire as to the explanation of the seeming paradox, or to condemn the cost figures as misleading.

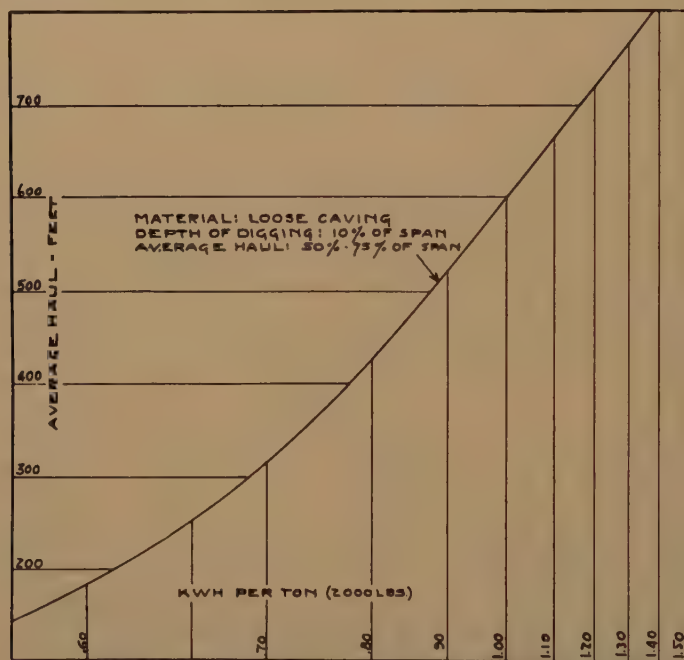


FIG. 14.—Power consumption for slackline cableway excavators. Kilowatt-hours (2000 lb.) for various average distances.

The principal explanation is that these machines are inherently a poor product for a manufacturing shop. They are too diverse and tend too readily to disintegrate into parts. They are a difficult product to merchandise. Inherently they do not consist of a few specific units or models. Too much engineering and the accompanying responsibility and risk are involved in the individual installation.

Some excavating and loading units, having reasonably fixed handling capacities and fixed unit costs, can be chosen without difficulty. The real economy of the job is very likely to be primarily concerned with the servicing of the excavating unit; that is, with the consideration of runways, haulage methods and costs, methods for spreading on spoil areas,

avoidance of lost time, wear and tear, obsolescence of haulage equipment, etc.

My point is that the over-all relative economy of this type of machine is inherently dependent upon the exercise of sound engineering judgment in the original selection and proper installation of the machine, and does not lend itself well to cut-and-try methods. In this respect long-range excavators may be said to be made to order for engineers on the producing side—both figuratively and literally.

APPLICATIONS

Sand and Gravel Excavation

The largest single use for the power scraper and/or the slackline cableway excavator has been in the production of sand and gravel.

The typical plant illustrated in Fig. 7 is a good example¹ of the usual power drag scraper installed for bank excavation for medium sized plants; i.e. scraper machines of from 60 to 200 hp. The drag scraper usually operates over distances of from 300 to 500 ft., with average hauls of about 0.6 times the radial distance or span. The output will usually equal and sometimes exceed the capacity indicated by Fig. 9.

The labor cost will just slightly exceed one man-hour per hour of operation. The maintenance cost will not exceed \$0.005 per ton per 100 ft. of average haul. Plant cost installed, without associated conveyors, if any, is likely to run from \$4500 for the 60-hp. machine to about \$15,000 for the 200-hp. machine. Over-all direct operating costs, including usual delays from shutdown of plant, etc., are likely to run from \$0.050 for the larger plant to \$0.085 per ton for the smaller. Similar plants operate on underwater excavation and noncaving coarse materials.

A typical application of the slackline cableway excavator for sand and gravel production is in the excavation in the Panama Canal zone of gravel from a bar in the river—turbulent at times—and the conveying and elevating of this gravel to a washing and screening plant high on the bluff over the river. The total span is 1500 ft. The motor is 100 hp. The bucket size is $1\frac{1}{2}$ cu. yd.

An objection sometimes raised to gravel excavation by these long-range excavators is inherent in these radial installations; namely, the irregular outline of the exhausted pits. Both manufacturers and users have overlooked a ready answer to this objection in the slight consideration that has been accorded to the tower machines, particularly to the general arrangement of Fig. 5. This machine operating on spans of about 200 ft. has large handling capacity and will load semipermanent haulage systems at a saving in cost under many situations. The deposit must be extensive and rather uniform, but the excavation can be extended indefinitely in rectangular units.

¹ Kenosha Sand & Gravel Co., Kenosha, Wis.; $2\frac{1}{2}$ -cu. yd. scraper, 100-hp. motor, 500-ft. span.

Stripping Overburden

The stripping of overburden from irregular hillside deposits of various industrial minerals is frequently handled to good advantage by power drag scrapers. Spans of 300 to 500 ft. are most frequent. An indispensable feature of such an installation is a rapid shifting device (Fig. 8) for controlling the digging point of the scraper. The flying block, which rides the back-haul cable, is shown in the foreground. The shifting cable, to which the flying block is attached, is preferably handled by a third drum at the hoist. Sometimes it is handled by a hand-power winch.

Frequently circumstances require the overburden to be deposited over a working face or similar obstruction, which favors the use of the low-end-dump cableway (Fig. 15). Such machines operate successfully over spans up to 500 or 600 ft. and permit the handling of overburden with a very small amount of labor and with a simple and relatively inexpensive plant.

Stripping of more regular and extensive deposits on level terrain is likely to be more favorable to overcasting the working face with revolving excavators, or for tower machines similar to Fig. 5.

Mucking and Tunnels

Hundreds of drag scrapers are in use in underground operations, for top-slicing, stoping, and for loading. Drag scrapers have been used in a limited way for mucking blasted material at headings and for taking excavated material away from power shovels in difficult stripping jobs.

Digging Clay

The excavation of clay for making cement or clay products is frequently handled to advantage by a power drag

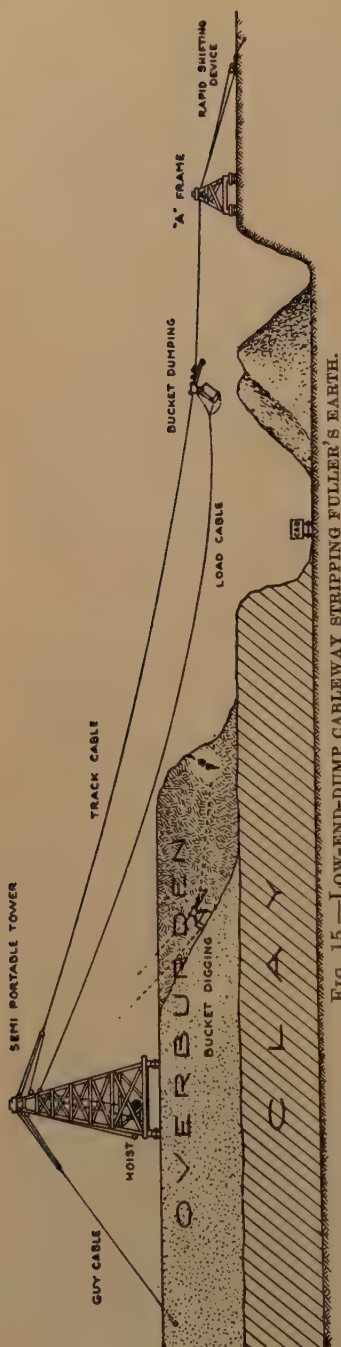


FIG. 15.—LOW-END-DUMP CABLEWAY STRIPPING FULLER'S EARTH.

scraper. An example is a 3-cu. yd. (125-hp.) drag scraper installed by Earls, Ltd., in England, for making cement. Clay is converted into slurry at the delivery point of the scraper, and pumped to the mill. A power-operated shifting device (Fig. 8) is indispensable to the plant. A better arrangement would be the use of the gravity-return scraper (Fig. 2). Rather stiff clays can be dug by a properly designed scraper.

The slackline cableway excavator is used when the dumping into elevated bins or hoppers is required.

The dragline cableway excavator is used for placer mining as well as for sand and gravel production—and not all placer mining is for metals. Two cableways have been used by the Bahia Corporation for the mining of black diamonds in Brazil on spans up to 800 ft. The motors are 100 hp. These machines would cost erected, under domestic conditions, about \$29,000 each.

Rehandling

While the problem of storing and reclaiming processed or partly processed materials is outside the subject of this paper, one application is particularly interesting in illustrating the value of the elevating and conveying feature of the slackline cableway. The Arundel Corporation, Baltimore, Md. takes sand and gravel to a washing and screening plant in drop-bottom scows. The scows dump their loads into a cushion pile in about 30 ft. of water alongside the plant. The cableway, a short-span $3\frac{1}{2}$ -cu. yd. machine, operates across the cushion pile and supplies the screening plant as required. This provides rehandling over a distance of about 250 ft. with one operator, and at the same time permits a quick turn-around of floating equipment.

Odd Jobs

One attractive feature of the drag scraper is its adaptability. It is used for some rather small, and occasionally odd jobs. One of these is the search for, and mining of, dumortierite, which is one of the rare minerals used in making the porcelain insulators used in spark plugs. An installation at the plant of the Champion Sillimanite Corporation, Detroit, is used to drag the ore over a grizzly for separation of fines, and then down a slide, which spreads it in a thin layer of dirt-free rock. Water is sprayed over the rock to bring out the vivid coloring of the dumortierite, which is then hand-picked, giving a recovery of over 95 per cent. The same scraper is used to dispose of the rock and dirt, which are dragged onto a waste dump.

The same plant has a scraper equipped with an attachment (patented) which permits it to turn a corner and operate around a dog-leg in a continuous back and forth motion. This dog-leg operation is accomplished with three haulage cables, and the third haulage cable is used for the rapid shifting device (Fig. 8) when the dog-leg operation is not required.

Hydraulic Stripping of a Stone Quarry

BY MARK SHEPPARD*

(Washington and College Park Meeting, October, 1937; New York Meeting, February, 1938)

DURING the winter of 1937, the writer visited a West Virginia stone quarry at which the overburden is stripped hydraulically. The quarry is in a bed of limestone, about 200 ft. thick, which outcrops on a hillside above a small river; the quarry face is 175 ft. high. The top of the stone is cut in all directions by erosion channels, which are filled with a tough, red clay. Some of the larger channels are 30 ft. deep and 8 ft. wide. The thickness of the clay overburden ranges from a few inches to 10 ft., the average being estimated as less than 3 feet.

The topography of the district is rugged, and the hillside above the quarry face is not suited to the use of the usual mechanical methods of moving dirt. Until hydraulicking was started 10 years ago, the overburden was dug by hand labor, dumped to the quarry floor and transported to the dump in quarry cars. Cleaning deep channels was a slow and expensive process.

When hydraulicking was started, the top of the quarry face was over 200 ft. above the level of the river. The pipe line was carried in a semi-circle around the top of the quarry about 200 ft. from the face and about 50 ft. above it. As there is no adequate supply of water above the quarry, the water must be pumped from the river. As quarrying progressed the pipe line was moved until the discharge points are now over 300 ft. above the river.

Water is pumped from the river by a four-stage centrifugal pump having a 5-in. intake and a 4-in. discharge. It is direct-connected to a 200-hp., 2300-volt, 1760-r.p.m. induction motor. At 1760 r.p.m. the rated capacity of the pump is 750 gal. per minute under a head of 600 ft. The pump is installed in a house of frame construction on the river bank, about 10 ft. above normal water level, which also houses the pump for the mill-water supply. Above the pumps are hoisting bays into which the motors are raised whenever the river reaches flood stage.

An 8-in. suction line, equipped with a strainer and a foot valve, takes the water from a concrete sump at the edge of the river. The pipe line from the pump consists of 1750 ft. of 8-in. spiral steel flanged pipe and

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800 ft. of 4-in. spiral steel flanged pipe. The connection from the pump to the 8-in. line is made by an 8 by 4-in. reducer followed by a check valve, a drain valve and a gate valve. About 100 ft. from the pump the line crosses under a railroad. A $22\frac{1}{2}^\circ$ and a 45° bend are used for this crossing. The line is laid on the surface on a horizontal curve of about 2° . In the first 1000 ft. it rises about 300 ft. and then becomes approximately level.

The 8-in. line contains five 8 by 8 by 4 tees, located at points convenient to the top of the quarry. The 4-in. line, 800 ft. long, is equipped with: one 8 by 4-in. reducer, four 4 by 4 by 4-in. tees, four 4-in. gate valves and one ball joint, which permits movement of the end of the 4-in. line through an arc of 30° without disconnecting the line.

The line has been in use approximately five months a year for 10 years. The cost of the unit, consisting of pump, monitors, pipe and fittings, and lighting equipment installed, was \$13,000. This excludes the cost of the pumphouse and concrete sump, which are charged to mill equipment.

When the change was made from hand stripping to hydraulicking the management was unable to obtain data from any similar operation on which to base estimates of volume and pressure required. In practice it has been found that using two 1-in. monitors reduces the pressure to such a degree that the streams do not cut the clay efficiently. When one 1-in. and one $\frac{3}{4}$ -in. monitor are used together the streams have much better cutting power. The increase in pressure of the stream when a single 1-in. monitor is used does not give enough increase in cutting power to compensate for the reduction in volume from the combination of 1-in. and $\frac{3}{4}$ -in. monitors. The usual practice, therefore, is to use one $\frac{3}{4}$ -in. stream for cleaning out channels and one 1-in. monitor for general cutting and sweeping. The 1-in. monitor has a 4-in. intake and is connected

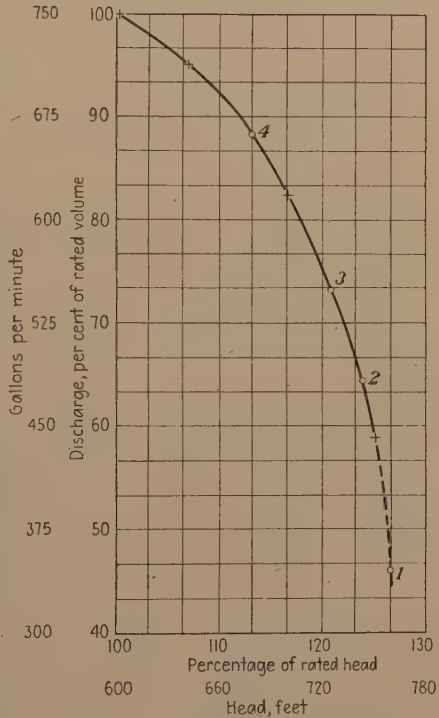


FIG. 1.—APPROXIMATE DISCHARGE OF CENTRIFUGAL PUMP WHEN WORKING AGAINST DIFFERENT HEADS.

1. Using one 1-in. monitor.
2. Using one 1-in. and one $\frac{3}{4}$ -in. monitor.
3. Using two 1-in. monitors. Substitution of 8-in. for present 4-in. pipe.
4. Using two 1-in. monitors.

directly to the 4-in. pipe line. The $\frac{3}{4}$ -in. monitor has a $2\frac{1}{2}$ -in. intake and is connected to the 4-in. line by about 20 ft. of $2\frac{1}{2}$ -in. rubber hose.

The waste is carried in launders across the quarry to a ditch, which carries it to the river. Although there are no legal restrictions on hydraulicking in West Virginia, the management has confined its hydraulicking to the period of high water between the latter part of October and the first of April.

Pressure-gauge readings taken at the pump were:

1. Using one 1-in. monitor..... 325 lb. = 750-ft. head
2. Using one 1-in. and one $\frac{3}{4}$ -in. monitor..... 318 lb. = 734-ft. head
3. Using two 1-in. monitors..... 310 lb. = 716-ft. head

The discharge curve in Fig. 1 was derived from test data of a centrifugal pump operating at approximately the same speed as the one in this unit.¹ The total heads and corresponding rates of discharge were reduced to percentages of rated capacity and the percentages plotted. For convenience the percentages in the figure are also expressed in feet of head and gallons per minute of this particular problem. Points 1, 2 and 3 are the working heads read on the gauge at the pump with 10 ft. of suction head added. Point 4 is the calculated result that could be obtained by the substitution of 8-in. pipe for the present 4-in. pipe and the use of two 1-in. monitors. Table 1 shows head and volume at monitor under different conditions of operation.

TABLE 1.—Head and Volume at Monitor

1 Size and Number of Monitors	2 Head at Pump		3 Quantity Dis- charged by Pump from Figure 1		4 Head Lost in Eleva- tion	5 Friction Loss		6 Head at Discharge		7 Calculated Cu. Ft. per Sec. Dis- charged by Monitors
	Lb. per Sq. In.	Head, Ft.	Gal. per Min.	Cu. Ft. per Sec.		8-in. Pipe	4-in. Pipe	Lb. per Sq. In.	Head, Ft.	
One 1-in.....	325	750	340	0.755	300	6.28	73.82	160	369.9	0.7551
One 1-in.....	} 318	734	480	1.066	300	11.56	128.96	127	293.5	} 1.064
One $\frac{3}{4}$ -in.....								126.5	292.1	
Two 1-in.	310	716	548	1.218	300	14.7	166.5	100.6	234.8	1.219
Two 1-in. with 8- in. line through- out.	290	670	660	1.467	300	30.2		147	339.8	1.467

¹ Kent Mechanical Engineer's Pocket Book, Ed. 9, 800, 734, 752.

CALCULATIONS

Friction in the pipe line is calculated from the formula:²

$$F = \frac{L}{d} \times \frac{4V^2 - 5V - 2}{1000}$$

in which F = friction head, ft.

L = length of pipe, ft.

d = diameter of pipe, in.

V = velocity of flow, ft. per sec.

The velocity is determined by the formula:

$$V = \frac{Q}{A}$$

in which V = velocity of flow, ft. per sec.

Q = flow, cu. ft. per sec.

A = cross-sectional area of pipe, sq. ft.

As a check on the volume of water used the discharge of the monitors is calculated from the following formula:³

$$Q = 8.02 CA\sqrt{H}$$

in which Q = cu. ft. water per second.

C = 0.91, a coefficient of nozzle discharge.

A = area of nozzle tip, sq. ft.

H = effective head at nozzle, sq. ft.

Example 1, Using One 1-inch Monitor:

Head at the pump, 750 ft.

Head lost in elevation, 300 ft.

Quantity delivered by pump, 340 gal. per min. = 0.755 cu. ft. per sec.

Friction loss from fittings:

In 8-in. line: One check valve = 32 ft. of pipe.

Two 8-in. gate valves = 32 ft. of pipe.

One $22\frac{1}{2}^\circ$ elbow = 8 ft. of pipe.

One 45° elbow = 12 ft. of pipe to be added to length of 8-in. pipe, 84 ft.

In 4-in. line: Four 4-in. gate valves = 38.4 ft.

One short-turn tee = 17 ft.

One 4-in. ball joint = 12 ft. to be added to length of 4-in. pipe, 67.4 ft.

² W. Cox: ref. 1, 734.

³ J. R. Thoenen: Sand and Gravel Excavation. U. S. Bur. Mines *Inf. Circ.* 6814, part 2, 81.

8-IN. PIPE

$$V = \frac{0.755}{0.3491} = 2.16 \text{ ft. per sec.}$$

$$F = \frac{1750 + 84}{8} \times \frac{18.64 + 10.80 - 2}{1000}$$

$$= 229.2 \times 0.0274 = 6.28 \text{ ft.}$$

$$750 - (300 + 73.82 + 6.28) = 369.9 \text{ ft. effective head} = 160 \text{ lb.}$$

4-IN. PIPE

$$V = \frac{0.755}{0.0873} = 8.65 \text{ ft. per sec.}$$

$$F = \frac{800 + 67.4}{4} \times \frac{299.29 + 43.25 - 2}{1000}$$

$$= 216.8 \times 0.3405 = 73.82 \text{ ft.}$$

$$\text{Discharge of giant } Q = 8.02 \times 0.91 \times 0.00545 \times \sqrt{369.9} = 0.7551$$

cu. ft. per sec.

Example 2, Using One 1-inch and One 3/4-inch Monitor:

Head at pump, 734 ft.

Head lost in elevation, 300 ft.

Quantity delivered by pump, 480 gal. per min. = 1.066 cu. ft. per sec.

Length of 4-in. line, 720 ft., with some loss from fittings.

8-IN. PIPE

$$V = \frac{1.066}{0.3491} = 3.05$$

$$F = \frac{1834}{8} \times \frac{37.2 + 15.25 - 2}{1000}$$

$$= 229 \times 0.0505 = 11.56$$

4-IN. PIPE

$$V = \frac{1.066}{0.0873} = 12.21$$

$$F = \frac{720 + 67.4}{4} \times \frac{596.3 + 61.05 - 2}{1000}$$

$$= 196.8 \times 0.6553 = 128.96$$

Head at discharge = $734 - (300 + 128.96 + 11.56) = 293.5$ ft. Head at the 3/4-in. monitor is reduced by 20 ft. of 2 1/2-in. hose. Interpolating in table of friction loss in hose (ref. 1, p. 752) shows friction loss in 2 1/2-in. hose to be 7.16 ft. per 100 ft. of hose. Then 20 ft. of hose would increase the friction loss 1.4 ft. Head at the 1-in. monitor would be 293.5 ft. and at the 3/4-in. monitor 292.1 ft.

$$Q \text{ discharged by } 3/4\text{-in. monitor} = 8.02 \times 0.91 \times 0.00306 \times \sqrt{292.1} =$$

0.382 cu. ft. per sec.

$$Q \text{ discharged by 1-in. monitor} = 8.02 \times 0.91 \times 0.00545 \times \sqrt{293.5} =$$

0.682 cu. ft. per sec.
1.064 cu. ft. per sec.

Example 3a, Using Two 1-inch Monitors:

Head at pump, 716 ft.;

Head lost in elevation, 300 ft.;

Quantity delivered by pump, 548 gal. per min. = 1.218 cu. ft. per sec.;

Length of 8-in. line, 1750 ft. plus 84 ft. for fittings;

Length of 4-in. line, 720 ft. plus 67.4 ft. for fittings.

$$\begin{aligned}
 & \text{8-IN. PIPE} \\
 V &= \frac{1.218}{0.3491} = 3.49 \text{ ft. per sec.} \\
 F &= \frac{1834}{8} \times \frac{48.72 + 17.45 - 2}{1000} \\
 &= 229.2 \times 0.0642 = 14.7 \text{ ft.}
 \end{aligned}$$

$$\begin{aligned}
 & \text{4-IN. PIPE} \\
 V &= \frac{1.218}{0.0873} = 13.95 \text{ ft. per sec.} \\
 F &= \frac{787.4}{4} \times \frac{778.40 + 69.75 - 2}{1000} \\
 &= 196.8 \times 0.8461 = 166.5
 \end{aligned}$$

Available head at monitors = $716 - (300 + 166.5 + 14.7) = 234.8 \text{ ft.}$
 Discharge of monitors at 234.8-ft. head:

$$Q = 8.02 \times 0.91 \times 2 \times 0.00545 \times 15.32 = 1.219 \text{ cu. ft. per sec.}$$

Fully 90 per cent of the friction head occurs in the 800 ft. of 4-in. pipe. If the 4-in. pipe were replaced by 8-in., a considerable increase in volume available at the monitor would result from the same expenditure of power.

The minimum head at which the stream cuts satisfactorily has been observed to be that which resulted from the use of one 1-in. and one $\frac{3}{4}$ -in. monitor; 293 ft. of head with the discharge of the 1-in. monitor was 0.682 cu. ft. per sec. = 307 gal. per minute.

Example 3b, Using Two 1-inch Monitors: Two 1-in. monitors at the minimum head required would discharge 614 gal. per minute. On the curve (Fig. 1), 614 gal. is shown to be discharged against a 700-ft. head.

Head at pump, 700 ft.; 10-ft. suction head = 690 ft. Head lost in elevation, 300 ft.

Quantity delivered by pump, 614 gal. per min. = 1.364 cu. ft. per sec.

Length of 8-in. pipe, 2550 ft. plus 148 ft. for fittings.

$$V = \frac{1.364}{0.3491} = 3.90 \text{ ft. per sec.}$$

$$F = \frac{2698}{8} \times \frac{60.84 + 19.50 - 2}{1000} = 337.25 \times 0.0783 = 26.40 \text{ ft.}$$

Available head at monitors would be

$$690 - (300 + 26.4) = 363.6 \text{ ft.}$$

which is far above the minimum requirement.

The curve shows that the pump will deliver 660 gal. per min., or 1.467 cu. ft. per sec. against a head of 680 ft. Subtracting 10 ft. suction head leaves 670 ft. head at the pump:

$$V = \frac{1.467}{0.3491} = 4.20 \text{ ft. per sec.}$$

$$F = \frac{2698}{8} \times \frac{70.56 + 21.00 - 2}{1000} = 337.25 \times 0.0895 = 30.18 \text{ ft.}$$

Available head at monitors is: $670 - (300 + 30.2) = 339.8$

$$Q = 8.02 \times 0.91 \times 0.0109 \times \sqrt{339.8} = 1.4669 \text{ cu. ft. per sec.}$$

Thoenen (ref. 3 and 4) estimates the carrying capacity of water used in hydraulicking gravel to be 3.7 to 12.6 per cent by volume.

STRIPPING

During the past winter, several factors made it desirable for the management to carry out an extensive stripping program, in the course of which over 2,000,000 tons of limestone were uncovered. The most important factor was that the quarry face had reached a point at which several deep channels as well as all of the surface could be washed directly to a launder there, making it possible to eliminate all hand cleaning of deep channels. As this condition might be changed by blasting the next round of holes, it seemed advisable to take advantage of the opportunity, even though it would require several years to quarry the stone exposed.

During about four months the stripping unit was operated 2465 hr. at a cost of \$9616.33. The breakdown of this cost is as follows:

Power.....	\$3,200.00
17,240 man-hours labor.....	5,331.33
Supplies, parts, etc.....	1,085.00
	\$9,616.33

The labor cost includes the wages of a foreman, but no superintendence or general overhead.

The estimated amount of dirt moved during this period is 8300 cu. yd. at a total cost of \$9616.33, or \$1.16 per cubic yard.

It would be practically impossible to cross-section the stripped surface of the stone accurately because of the prevalence of channels of varying size, so the estimate of volume of dirt moved is based on the solids contained in the launder water.

From a number of samples it is estimated that the launder water carries 3.75 per cent, by weight, of solids. The clay in place is estimated to weigh 90 lb. per cubic foot.

The exact volume of water used was not determined, but it is estimated that one 1-in. and one $\frac{3}{4}$ -in. monitor were used during 75 per cent of the operating time and that during the other 25 per cent of the time one 1-in. monitor was used alone. On this basis 3500 cu. ft. of water was used per operating hour.

Pounds of dirt handled per hour equals:

Cubic feet of water $\times$ weight of water per cubic foot $\times$ percentage of
dirt carried;

or $3500 \times 62.5 \times 0.0375 = 8203$ lb. per hour

$\frac{8203}{90} = 91.1$ cu. ft. per hour = 3.37 cu. yd. per hour

⁴ J. R. Thoenen: Sand and Gravel Excavation. U. S. Bur. Mines *Inf. Circ.* 6875, part 5, 219.

In terms of dirt in place, the percentage by volume of dirt carried in the launder water is:

$$\frac{91.1}{3500} = 2.6 \text{ per cent}$$

Table 1 shows that the use of 8-in. pipe throughout the whole line would have made 1.467 cu. ft. per sec. available at a head of 339 ft. This equals 5281 cu. ft. of water per hour, capable of carrying 137.4 cu. ft., or 5.09 cu. yd. of dirt per hour.

At 90 per cent efficiency, 4.58 cu. yd. would be moved per hour with no increase in cost for power or labor. The hourly cost of operation, \$3.90, would not be materially changed, and cost per cubic yard would be \$0.851.

DISCUSSION

(A. C. Fieldner and J. Murray Riddell presiding)*

O. BOWLES,† Washington, D. C.—The figure of \$1.16 a cubic yard seems very high for hydraulic stripping. Of course, the top of the quarry is 300 ft. above the river and the stripping averages less than 3 ft. thick, and there is great difficulty in getting clay out of tortuous passages in eroded limestone surfaces. Sometimes stripping costs are extremely low. With a smooth surface, plenty of water, and a place not too high above the river level, where material may be washed to a convenient settling basin, the cost may be only a few cents per cubic yard. Perhaps Mr. Fulton will tell us about costs in Florida.

C. A. FULTON,‡ Baltimore, Md.—Conditions in the phosphate field in Florida are not comparable with this operation. We have a much thicker bed of matrix, which is the part of our operation that is hydraulic. We use guns with 2-in. nozzles with about 200 lb. pressure per square inch at the nozzle, two guns in a pit, throwing about 4000 gal. per minute, and we wash down about 300 yd. per hour of matrix. It flows only a short distance, say 100 ft., to an improvised sump that we have to construct in the floor of the pit from time to time as the face gets too far away from the guns when we move up later to another location of the same kind.

We pump the matrix with 12-in. single-stage pumps, sometimes as many as three in the line including the pump in the pit, with 4000 gal. a minute of water. That, I think, figures out to 240,000 gal. an hour, roughly about 30,000 cu. ft., or roughly about 1100 cu. yd.; that is, 1100 cu. yd. of water to about 300 yd. of solids.

Our costs are very much lower than Mr. Sheppard mentions. They would have to be in order to produce phosphate—a low-priced commodity. The out-of-pocket cost, per cubic yard including the hydraulicking, the high-pressure stationary pump, and the pumping of the pulp to the washer, is about nine cents.

MEMBER.—Is the waste allowed to go back into the river?

M. SHEPPARD.—It is allowed to go back into the river.

* Mr. Fieldner presided at the Washington Meeting, Mr. Riddell at New York. The discussion at the two meetings has been combined.

† Chief, Building Materials Section, U. S. Bureau of Mines.

‡ President, Southern Phosphate Corporation.

G. S. RICE,* Washington, D. C.—One of my investigations when I first went to work with the Bureau of Mines 30 years ago was in connection with the California Debris Commission, to study what might be done to prevent the spoiling of the lands below by hydraulicking in the mountains. It seems to me that it should be and in most countries is against public policy to turn clay and sand, or any debris, into the streams. Part of the cost that ought to be included in this study should be the cost of preventing the clogging of our streams, which causes damage from subsequent floods and makes the water foul. In some instances, it may not lead immediately to serious damage, but later on there is filling behind dams, trouble in the spreading of the debris over the lands below, and so forth. Often it can be avoided. About 35 years ago, before the Bureau of Mines was started, I was brought into a case in Florida, in stripping operations to uncover gravel phosphate beds, and under the particular conditions was able to work out a system much cheaper than that of hydraulicking the overburden or removing it by steam shovel, the systems that were being used, by employing a movable crane with drag scraper. After the first cuts were made through the gravel phosphate, the overburden was dropped into the excavation by the crane, which moved on top of the gravel phosphate. I think that nowadays, as a public policy, we should not fill our streams with debris.

MEMBER.—Have you any comparable costs for hand stripping?

J. R. THOENEN,† Washington, D. C.—At the quarry that Mr. Sheppard is discussing, the operators have never done a great deal of hand work in stripping. As pointed out, the surface erosion cavities are large, some as much as 12 or 15 ft. wide. I have some records in my office of the cost of hand stripping under more or less similar conditions, which run as high as \$1.50 and \$2.00 per cubic yard. This high cost is due to the necessity of shoveling the filling by hand from wide and deep cavities to the surface, where it can be reached by mechanical equipment. This type of operation is similar to digging test pits by hand.

In some instances where the cavities are large enough, clamshell cranes have been used but they are not entirely satisfactory because of the irregular shape of the erosion pockets. Even with these the final cleaning must be done by hand.

At the quarry under discussion, the management has found it more economical to clean the cavities and surface in one hydraulic operation rather than supplement hand work by hydraulic giants. This accounts for the relatively high cost for hydraulic stripping.

The \$1.16 per cubic yard given as the actual cost of removing the overburden by hydraulicking, is of course, high compared with western practice, but the erosion cavities in the surface of the stone with which the operators had to contend must be remembered.

S. S. SMITH, New York, N. Y.—In most states there are Efficiency Game Commissions that will not allow the debris to go into a body of water—or a State Commission governing the disposal of the waste. Are you allowed to wash this into the river without any restraint whatsoever?

J. R. THOENEN.—I understand there was no objection as long as the work was done in the winter.

G. S. RICE.—What is the possibility of a settling basin in the valley?

J. R. THOENEN.—At that particular quarry there is no possibility of a settling basin near the quarry.

* Consulting Mining Engineer.

† Nonmetal Mining Section, U. S. Bureau of Mines.

Moreover, this quarry has stripped, in four-months operation, sufficient stone to last for possibly 15 or 20 years, so there will be no further stripping of this quarry.

H. I. SMITH,* Washington, D. C.—Two factors involved in hydraulic stripping would seem not to have been discussed in this paper; namely, nozzle pressure and carrying capacity of the water after it leaves the nozzle. Nozzle pressure must be known if cutting power is to be determined; and, as the carrying capacity of water varies with the cube of its velocity, the low carrying capacity mentioned by the author may have been the result of low velocity of the transporting water rather than insufficient solid material to load the stream.

M. SHEPPARD.—Of course the surface is very irregular and the clay is tough, so that after it is cut more water is required to transport it, because of the rough surface.

As to the nozzle pressure, using the 1-in. monitor, 369 ft. of head, or 160 lb. pressure at the nozzle; using the 1-in. and $\frac{3}{4}$ -in. monitor, the pressures were 127 and 126½ lb. for the two different nozzles. That apparently cut the clay as well as, or nearly as well as, the higher pressure. The nozzle pressure that did not cut the clay satisfactorily was 100 pounds.

* Chief, Mineral Leasing Division, U. S. Geological Survey.

Drilling and Blasting Practice of Consolidated Quarries Corporation

BY NELSON SEVERINGHAUS,* MEMBER A.I.M.E.

(Washington and College Park Meeting, October, 1937; New York Meeting, February, 1938)

THE Rock Chapel plant of Consolidated Quarries Corporation (Fig. 1) is three miles northeast of Lithonia, DeKalb County, Georgia. It was opened about eight years ago for crushed stone aggregate. This paper reviews a number of experiments that have been tried in drilling and blasting, and gives the practice in use today. It also presents several theories on the action of explosives in this type of blasting.



FIG. 1.—PLANT OF CONSOLIDATED QUARRIES CORPORATION, WITH QUARRY AT UPPER LEFT.

The rock worked is a bare dome of granite gneiss about 3000 ft. in diameter, rising 150 ft. above the surrounding ground (Fig. 2). It is uniform in composition and quality, without seams, and extends downward to unknown depth below the quarry floor. A water well drilled 600 ft. remained in the same rock. The principal difficulty encountered in blasting is an easy split, which tends to break down the rock in large blocks.

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* Superintendent, Consolidated Quarries Corporation, Lithonia, Ga.

PRIMARY DRILLING PRACTICE

First work was with tripod-mounted air hammers drilling 20-ft. vertical holes spaced about 8 by 10 ft. All rock was removed by this method until the face was about 60 ft. high. At this height, breaking was done in three 20 by 20-ft. benches, each bench having two rows of holes spaced 8 ft. apart. These were loaded with 60 per cent quarry gelatin dynamite and exploded with instantaneous No. 6 electric blasting caps. Some toe holes were used to help the bottom bench maintain a level floor. Four Ingersoll-Rand X-71 tripod-mounted drifters produced an average of 135 tons of rock per hour. About six Jackhammer drills were kept busy on secondary drilling for breaking up blocks on the floor.

Because of the hazards of such work, the high cost of clearing off benches for drilling, and the relatively small amount of rock that could be broken down at one time, a traction mounted, electrically powered blast-hole drill for 6-in. holes was then put to work on top of the 60-ft. face. This drill averages about $2\frac{1}{2}$ ft. of hole, or 100 tons of rock drilled, per hour, including moving and bit-changing time. As the crushing plant is now taking about 200 tons per hour of operating time on a single shift, the drill is run two shifts to keep drilling ahead. A driller and helper on each shift operate the drill and hand-sharpen the 6-in. chisel bits, with a coke forge for heating. A 6-in. ring of Jackhammer holes about 2 ft. deep is drilled before each hole is started, to aid in collaring. These are not blasted, because the well

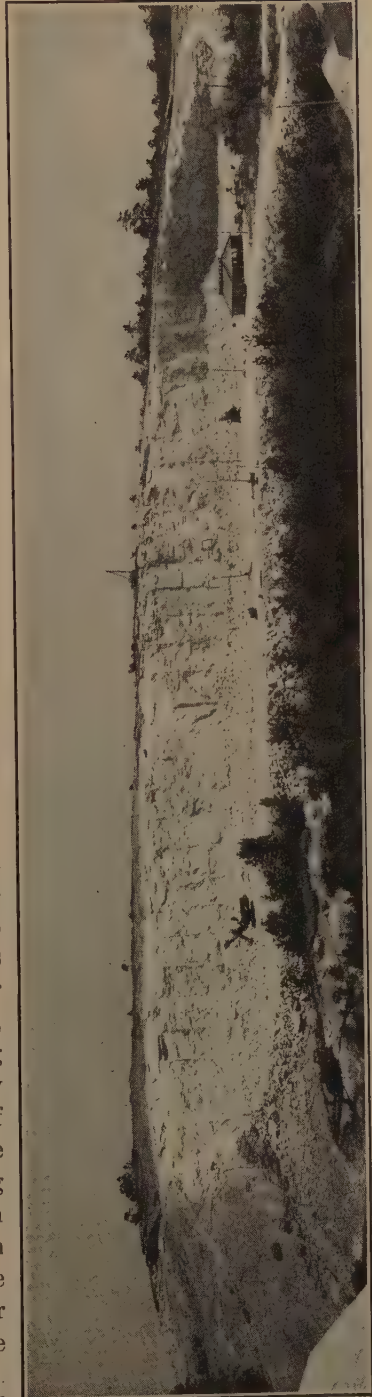


FIG. 2.—QUARRY AS SEEN FROM CRUSHER BUILDING.

drill quickly breaks out the remaining core without blasting. A series of elevation bench marks is occasionally established back from the rim and drillers use a hand level to determine hole depth from these marks.

Holes are drilled 5 ft. below the quarry floor, so that shovels can maintain a level floor on broken stone. As soon as a hole is drilled, it is covered with a slab of rock sealed around the edges with concrete, and the depth is marked on the concrete. This effectively excludes loose material from washing in and filling up part of the hole. When loading time comes, such caps are readily knocked off. An air-displacement cylinder with bottom check valve is lowered into each hole to remove water just before loading. This device clears out the water much faster and with less labor than a hand bailer.

The spacing, burden and depth of each hole are checked before loading and the tons that will be moved by each is figured. This, divided by the loading factor decided upon, gives the amount of dynamite to be placed in each.

PRIMARY BLASTING PRACTICE

Placing of holes for the first shot in June 1930 followed usual practice for this height of face with average spacing of 17 ft. and burden of 21 ft. The 24 holes of the first shot were loaded with 4000 lb. of 75 per cent quarry gelatin in the bottoms and 7100 lb. of 60 per cent quarry gelatin above. A considerable amount of screenings stemming was used, to maintain the ratio of 4 tons to 1 lb. decided upon and yet raise part of the load up toward the top of holes. Detonation was by Cordeau fuse in each hole connected to a Cordeau trunk line on top with an electric cap on one end. Fragmentation was poor (Fig. 3) and an excessive amount of secondary blasting was necessary to prepare the stone for shovels (about twice that previously required for tripod-drilled rock). This indicated a reduction of burden and spacing. As the next shot was already largely drilled, the spacing was kept the same but the burden was reduced by taking off a slab with tripod drills, so that it averaged but 15 ft. The loading basis was changed from 4 to $3\frac{1}{2}$ tons of rock to 1 lb. of dynamite. The dynamite used was about one-half 75 per cent and one-half 60 per cent quarry gelatins. Results were much better than those of the first shot and breakage was comparable with that of tripod-drill shooting.

Holes have never been sprung here. Column loading gives a better distribution of explosives behind the rock to be moved and effectively breaks a clean face to the quarry floor level.

On the next several shots, an attempt was made to get good breakage by using a lower loading factor of about 3.3 tons per pound with 18-ft. spacing and 20-ft. burden, to keep drilling costs per ton from running too high. This gave solid column loading with no stemming and was about the maximum load that could be used without throwing rock beyond the

quarry floor. In shot No. 4, semigelatin ammonia dynamite was substituted for 60 per cent quarry gelatin, with some saving in cost and no appreciable change in effect. Fragmentation remained none too satisfactory.



FIG. 3.—POOR FRAGMENTATION OF SHOT NO. 1.



FIG. 4.—WELL BROKEN PRIMARY SHOT.

With the idea that an explosive having higher velocity might aid fragmentation, shot No. 5 was made largely with 60 per cent straight nitroglycerin dynamite, and there was some improvement in breakage.

Experiments with the next three shots indicated that a wider spacing, 24 ft., and lowered burden, 17 ft., gave good fragmentation even when less costly and less dangerous semigelatin and ammonia dynamites of

slower speed were used. For some time, spacing was kept at this figure and increasing percentages of bulky ammonia dynamite were used until the loading ran about one-fourth of 60 per cent semigelatin and three-fourths of 45 per cent bulk-strength ammonia dynamites. Then spacing and burden were again experimented with until the present standard of 32 by 16 ft. was reached. Spacing wider than 32 ft. makes it impossible to break out all of the toe clean; decreased spacing gives poorer fragmentation. Further decrease in burden does not seem to help breakage, and increase hinders it.

On shot No. 17 the charge was detonated by an instantaneous electric blasting cap placed near the bottom of each hole instead of by Cordeau trunk line. This gave better breakage and distribution of rock on the quarry floor than any previous shot.



FIG. 5.—FACE AFTER SHOT NO. 35, SHOWING POT HOLES BETWEEN CAP LOCATIONS.

Failure of five holes in shot No. 19, which was connected in the same way, showed that well insulated wire of ample size was necessary. Use of long-leg wire (125 ft. No. 18 duplex enamel wire) caps improved convenience and safety. On the next shot, the Cordeau trunk line, which had been used with caps as a safety measure, was omitted. Tests on the ground indicated that the fifth hole would cut the Cordeau trunk line before its wave of detonation had reached this point and thus it could not be counted on to carry out the shot in case of cap failure further on. This test duplicated the usual Cordeau blast-hole hook-up, except that the Cordeau was laid out on the ground. Whenever initiation was in a right-angle branch line the trunk line would be cut at the junction point without being exploded. Observation of distances indicates that the explosion would reach the fifth-hole junction point first through the branch from its cap at the hole bottom and thus the trunk line would be cut there.

The next step in detonating methods was the use of two, and then three, electric blasting caps in each hole. This further improved break-age. "Pot holes" of highly fractured rock about halfway between cap locations appeared regularly in the face after such shots and indicated an increased effect of explosives at points where detonation waves met (Fig. 5).

PRESENT METHODS OF BLASTING

This method of shot initiation was followed until Nitramon, the new safety agent for quarry blasting with deep well-drill holes, was intro-

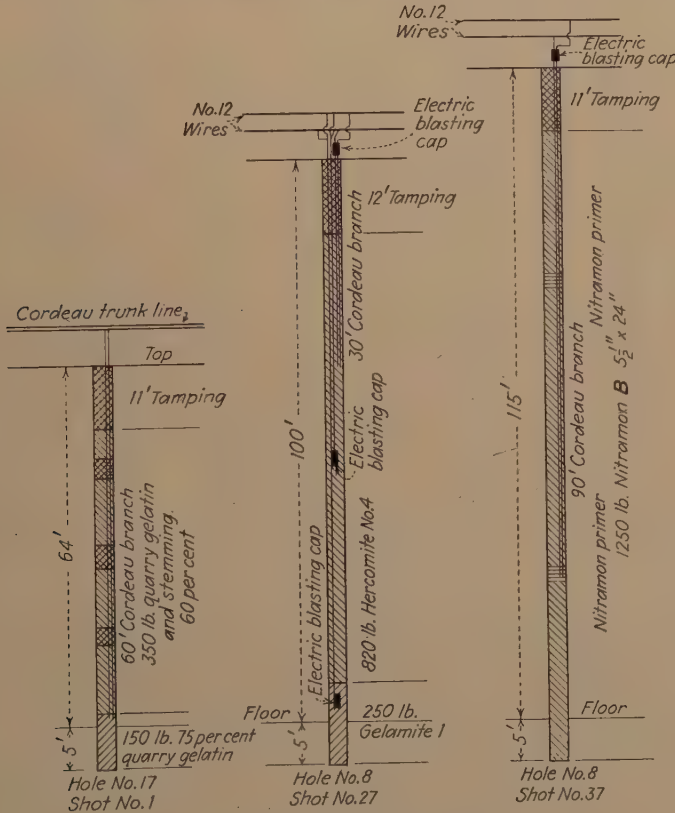


FIG. 6.—TYPICAL HOLES.

duced. To fully utilize the safety features of this agent, it was necessary to eliminate the use of caps in holes. The facts that Cordeau is faster than Nitramon and will not initiate detonation of Nitramon without a special primer were suggested by Mr. R. C. Crumbaugh, of the duPont company, as making possible the combination of the advantages of

multiple initiation points within the hole and the safety of the Cordeau-Nitramon method of shooting. This was tried very successfully in the last two shots. A chart of typical holes in shots 1, 27 and 38 is shown in Fig. 6, to illustrate the various methods that have been used to initiate detonation. With Nitramon, an instantaneous electric blasting cap is used on the Cordeau leading down into each hole instead of a Cordeau trunk line, as the instantaneous initiation of all holes gives better results. Although the time interval from one end of a shot to the other is small with Cordeau (speed about 17,500 ft. per sec.), it is significant in this rock. Fig. 7 illustrates my theory for this difference, which results seem

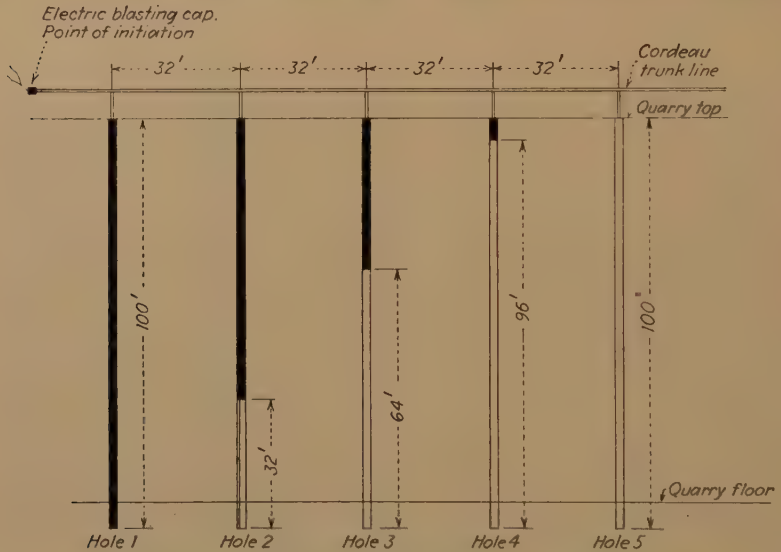


FIG. 7.—ONE END OF CONVENTIONAL QUARRY SHOT.

Darkened areas represent exploded depths at time first hole is entirely detonated.

to justify. This illustration is drawn to represent conditions of a Cordeau trunk-line connected shot at the time the detonation wave has reached the bottom of the first hole. Darkened hole areas represent depths already detonated. As the holes are 32 ft. apart, detonation at this instant will be 32 ft. from the bottom of hole No. 2; 64 ft. from the bottom of hole No. 3; just starting in No. 4, and not yet have reached No. 5. Thus the effect is that of a wedge being driven in from the upper left-hand corner. In a rock as easily split as that of Rock Chapel, this perhaps allows the expenditure of some of the explosive energy in most holes out in a crack already started, instead of in breakage of solid rock in front of it. Frequent excellent breakage at the end of shots here tends to confirm this theory.

A lowering of secondary shooting costs after the change from Cordeau to cap detonation further substantiates this idea.

TYPICAL LOADING CHARTS

Loading charts of shots 1, 27 and 38 are given in Tables 1 to 3, to show details of the major changes in drilling and blasting described above.

TABLE 1.—Loading Chart of Shot No. 1^a

Hole No.	Depth, Ft.	Spacing, Ft.	Burden, Ft.		Tons	Tons 4	Total Explo- sive, Lb.	75 Per Cent Quarry Gela- tin, Lb.	60 Per Cent Quarry Gela- tin, Lb.	Tamp- ing, Ft.
			Top	Bottom						
1	56	19	10	16	1,030	257	250	100	150	10
2	58	18	19	28	1,820	455	450	200	250	11
3	61	19	20	27	2,010	503	500	200	300	10
4	63	18	20	26	1,990	498	500	250	250	13
5	62	18	21	24	1,960	490	500	200	300	12
6	63	18	20	26	1,980	495	500	200	300	11
7	64	18	20	26	2,040	510	500	250	250	12
8	65	18	20	24	1,980	495	500	150	350	12
9	66	18	20	24	2,020	505	500	200	300	11
10	66	18	19	22	1,840	460	450	100	350	11
11	69	19	20	25	2,170	543	550	150	400	10
12	65	16	21	26	1,850	463	475	200	275	13
13	69	16	23	26	2,040	510	500	200	300	12
14	69	16	23	25	1,940	485	500	150	350	11
15	65	16	22	26	1,840	460	450	200	250	12
16	69	16	23	25	1,990	498	500	200	300	12
17	69	16	22	25	1,980	495	500	150	350	11
18	69	16	22	24	1,970	493	500	150	350	11
19	67	16	22	24	1,860	465	475	100	375	10
20	66	16	22	24	1,880	470	475	150	325	11
21	65	16	22	24	1,670	418	425	150	275	11
22	66	16	22	24	1,730	433	450	200	250	10
23	65	13	22	24	1,810	453	450	150	300	10
24	65	Cutoff	14	16	1,125	281	200		200	10
390					44,525		11,100	4,000	7,100	

^a Rock Chapel plant, June 11, 1930. All holes 6-in. dia. Granite; weight 164 lb. per cu. ft. All holes drilled 5 ft. below quarry floor. Tonnage shown only available stone or that which would be recovered. Loading basis: 4.0 tons to 1 lb. of dynamite. Percentage of strengths used: 75 per cent quarry gelatin, 36 per cent; 60 per cent quarry gelatin, 64 per cent.

Results: (1) Poor fragmentation; (2) toe pulled clean; (3) no damage done; (4) clean quarry face, very little back break.

Next Shot: Use 50 per cent of 75 per cent quarry gelatin. Reduce burden. Increase loading basis to 3.8.

For secondary drilling of blocks on the quarry floor, six to eight light-weight, hand-held air hammers are used. On about 80 per cent of the

blocks, holes 2 ft. deep or less are needed, and they are drilled with 40-lb. hammers using $1\frac{1}{4}$ -in. cross bits on $\frac{7}{8}$ -in. hollow hexagon steel. The particular drills used for this have been found to drill about 20 per cent faster without shank collars, so none are used. Where deeper holes are needed, 1-in. steel and heavier hammers (50 lb.) are used. Starting bits are $1\frac{7}{8}$ -in. diameter with $\frac{1}{8}$ in. change for each 2 ft. depth.

Most of the dynamite used for this breaking is semigelatin packed in 1 by 4-in. cartridges. On deeper holes, $1\frac{1}{4}$ by 8-in. of the same grade are used. All holes are primed with 4-ft. iron-wire electric blasting caps and tamped with stone screenings. Iron wires are used for economy, the cost being less than for copper-wire caps of the same length. They have been entirely satisfactory in this kind of work. About 30 caps are connected in series with No. 20 annunciator wire, and such groups paralleled on the end of No. 18 duplex all-rubber cable. Final connections are made at the end of the working shift and firing is done from 550-volt power line at a point in a crusher building overlooking the whole quarry. Failures with these methods are extremely rare, although as high as 300 blocks have been shot at one time. Time saving and greater safety justify the higher cost of electric caps compared with fuse and cap shooting.

TABLE 2.—*Loading Chart of Shot No. 27^a*

Hole No.	Depth, Ft.	Spacing, Ft.	Burden, Ft.	Tons	Tons 3.5	Total Explosive, Lb.	Gela- mite 1, Lb.	Herco- mite 3	Caps, Ft. from Top	Tamp- ing, Ft.
1	92	29	14	3,060	875	870	270	600	85 48	16
2	97	29	16	3,700	1,060	850	280	570	95 45	12
3	100	29	16	3,800	1,090	900	250	650	90 49	14
4	105	30	15	3,880	1,110	1,080	250	830	100 55	14
5	107	32	15	4,220	1,200	1,050	250	800	100 50	12
6	107	32	16	4,480	1,280	1,120	250	870	100 58	12
7	107	32	12	3,360	960	720	250	470	100	14
8	105	33	15	4,270	1,220	1,070	250	820	95 55	12
9	100	33	13	3,520	1,000	1,000	250	750	90 58	12
10	112	31	15	4,270	1,220	1,160	250	910	95 54	14
11	112	32	15	4,400	1,260	1,000	150	910	99 54	13
12	112	32	15	4,400	1,260	1,070		1,070	103 57	14
13	110	32	15	4,330	1,240	1,050		1,050	95 50	13
14	112	31	16	4,550	1,300	1,000		1,000	97 54	15
15	112	30	14	3,860	1,100	1,050		1,050	98 51	12
Total.....	1,590	467	222	60,100	17,175	15,050	2,700	12,350		
Average.....	106	31	14.7	4,006	1,145	1,003				

^a Rock Chapel plant, Jan. 10, 1936. Net recoverable tonnage, 57,000. All caps in parallel on No. 12 W. P. wire. Cordeau in each hole but no trunk line.

Results: Good breakage and distribution. Toe pulled out clean. No damage done. Top left in good shape; little back break.

CONCLUSIONS

Conclusions reached from our experience discussed above are:

1. Although conventional hole spacing is a good guide in starting out with any new quarry, experiments may show an unconventional spacing to give much better results in some rocks. At Rock Chapel, 32-ft. spacing and 16-ft. burden gives best results.

2. Spacing is much more important in securing good fragmentation in some quarries than type and grade of explosive used. No explosive has yet given good fragmentation here when burden exceeds spacing.

3. Instantaneous detonation of all holes gives better fragmentation than is secured with Cordeau trunk-line detonation.

TABLE 3.—*Loading Chart of Shot No. 37^a*

Hole No.	Depth, Ft.	Spacing, Ft.	Burden, Ft.	Tons	Calcu- lated Cans Nitra- mon	Actual Load				Primers		Tamp- ing, Ft.
						Nit. B	Nit. C	Primers	Total			
1	116	32	16	4,870	52							"
2	118	32	15	4,640	52	48	2	2	52	45	95	11
3	118	32	15	4,640	53	48	2	2	52	44	88	11
4	120	32	16	5,030	54	48	2	2	52	40	90	11
5	120	32	15	4,720	54	50	2	2	54	38	90	10
6	120	32	16	5,030	54	49	2	2	53	40	90	11
7	117	32	15	4,900	53	48	2	2	52	38	82	12
8	120	32	15	4,720	54	50	2	2	54	38	90	11
9	117	32	16	4,910	53	48	2	2	52	33	87	11
10	117	32	15	4,600	53	47	2	2	51	42	86	11
11	115	32	15	4,530	51	47	2	2	51	31	79	13
12	110	32	16	4,620	49	44	2	2	48	40	84	10
13	112	32	16	4,700	50	41	3	3	47	36	80	18
14	110	31	17	4,750	49	46	2	2	50	44	82	12
15	100	32	17	4,460	45	38	2	2	42	36	80	12
16	95	32	16	3,990	42	37	2	2	41	31	71	13
Total...	1,825	511	251	75,110	819	689	31	31	751			
Average.	115	32	16	4,694								

^a Rock Chapel plant Aug. 6, 1937. Hole No. 1 could not be loaded because of offset from previous shot. At each hole an electric blasting cap was placed on a branch line of Cordeau, using special unions. All caps connected in parallel to No. 12 R.C. wire.

Results: Excellent fragmentation. Toe pulled clean. Little back break. No damage.

Summary: 689 cans Nitramon B @ 25 lb. 17,225 lb. ($5\frac{1}{2} \times 24$)
 31 cans Nitramon C @ 22.22 lb. 690 lb. ($5\frac{1}{2} \times 24$)
 31 primers @ 22 lb. 682 lb. (5×24)
 Total 18,597 lb.

Total rock: 15 holes, 70,240 tons. Loading factor, 3.77 tons to pound.

4. Several separated points of detonation within a hole help fragmentation.

5. The safety of Cordeau can be combined with instantaneous firing of all holes by using electric blasting caps on top of Cordeau in each hole. This keeps all caps off the job until all loading is completed.

6. Separated points of detonation within each hole are secured by using Cordeau with the new blasting agent, Nitramon, and separated primers.

DISCUSSION

(Chester A. Fulton presiding)

S. L. WINDES,* College Park, Md.—Are the primer cartons you use now of the same physical dimensions as the Nitramon?

N. SEVERINGHAUS.—The Nitramon is put up in tin cans, and so are the primers. The primers carry a soldered tunnel along the side and we use cans about as large as the holes will stand. The holes are 6-in. and the Nitramon cans are 5½. With the tunnel on the side, a 5-in. primer can is used, and the Cordeau is strung up through the tunnel.

S. L. WINDES.—If there were a misfire, could the primer cartridge be set off by the shovel?

N. SEVERINGHAUS.—The primer itself is quite insensitive. I think the primer cans are about one-half T.N.T. and the other half inside the can is the Nitramon material. I do not believe the T.N.T. of itself could be set off by a shovel running against it.

J. W. KOSTER,† Wilmington, Del.—It is certainly more sensitive than the Nitramon, but it is quite an insensitive thing. We recently made some tests by shooting into the T.N.T. of a Nitramon primer a 30-30 bullet. Out of 10 trials we had no detonations.

For how many years have you been shooting with a 16-ft. burden and 32 ft. between holes?

N. SEVERINGHAUS.—We standardized on that about four years ago, I think. We decided we had reached about the best spacing, and decided on that.

J. W. KOSTER.—And what difference has that made in the secondary blasting as compared, for instance, with your old standard of 20 by 20 spacing?

N. SEVERINGHAUS.—In amount, I think it has reduced the secondary blasting by at least 50 per cent. In cost, the bigger effect is in speeding up shovels, because shots such as those put down will, in places, be at least 20 ft. deep in broken rock, therefore any large rock has to be first uncovered by the shovel before it can be broken up, which greatly slows down the operation of a shovel.

J. W. KOSTER.—With the old 20 by 20 spacing, did you frequently have trouble with toe?

N. SEVERINGHAUS.—Yes. We had to do a considerable amount of jackhammer work on toe, and this new spacing has practically eliminated the toe trouble.

* U. S. Bureau of Mines.

† E. I. du Pont de Nemours & Co.

Quarry and Plant of Reliance Rock Asphalt Corporation

By E. H. CRABTREE, JR.* AND W. F. NETZEBAND,† MEMBER A.I.M.E.

(New York Meeting, February, 1935)

THE productive area of asphalt-bearing sandstone in Missouri is near the Missouri-Kansas state line near Nevada, the county seat of Vernon County about 100 miles south of Kansas City. While production thus far has been confined to Vernon and Barton counties, there are known deposits in adjacent counties. The area is served by four railroads: the Missouri Pacific, the Kansas City Southern, the Missouri-Kansas & Texas, and the St. Louis & San Francisco.

The plant of the Reliance Rock Asphalt Corporation is in Vernon County, about 5 miles west of Nevada, at Ellis, a flag station on the Missouri, Kansas & Texas Railroad. A siding has been built to the plant so that the finished product can be loaded directly into railroad cars. The property consists of a lease of 240 acres in one block. All portions of the property not required for quarrying or processing the asphalt-bearing sandstone are used for farming by the fee owner. The entire tract is very flat so that there are no transportation difficulties from quarry to plant.

GEOLOGY

The commercial asphalt-bearing sandstones are all of Pennsylvanian age, the major portion of the production coming from the Cherokee formation, the basal member of the Des Moines group.

The Cherokee formation includes all beds between the base of the overlying Fort Scott limestone and the Mississippian limestones below. It underlies most of the area, but in the west and northwest townships of Vernon County, and elsewhere in local outliers, the Henrietta limestone, forming mounds, escarpments and high ridges, covers the Cherokee, and is in turn covered by the Pleasanton shale. In the eastern portion of the counties, the Mississippian limestones are exposed. The thickness of the Cherokee varies according to the amount that has been removed by surface erosion. Where fully present under the Henrietta; it is approximately 375 ft. thick, but over most of the county it ranges from 100 to 200 ft. thick.

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† Mining Engineer; Statistician, Tri-State Zinc and Lead Ore Producers' Association, Joplin, Missouri.

The formation consists chiefly of shale but contains a considerable quantity of sandstone, thickest and most abundant in the lower part, but also in considerable amount interstratified with the upper shales. The sandstones vary from thin seams to beds over 60 ft. thick composed chiefly of fine to medium, angular quartz grains, gray to brown in color and commonly micaceous. They vary greatly in hardness, though generally they are not tightly cemented, the most common cementing material being calcium carbonate or iron oxide.

They are lenticular, thickening and thinning rapidly and irregularly, and laterally grading into shale. For this reason it is difficult to correlate individual beds over any great distance, although locally an individual bed may be persistent.

The Henrietta formation and the overlying Pleasanton shale contain asphalt-bearing sandstones, but as no production has been made from them in the area under discussion, they will not be described in detail.

Asphalt-bearing Sandstone Deposits

The asphalt-bearing sandstones are found at two general horizons in Barton and Vernon counties; near the base of the Cherokee formation in the Clear Creek sandstone (Bartlesville) and higher up in a sandstone that may be the equivalent of the Red Fork sand of Oklahoma. This upper sandstone is the productive sandstone at the plant under discussion.

The lower sandstone has been found impregnated with asphalt for a thickness of 30 ft., but the usual thickness of the asphalt-bearing portion is under 20 ft. The upper sandstone is impregnated with commercial quantities of asphalt for a thickness of 10 to 15 ft., although there is often an overburden of 5 to 10 ft. of lean asphalt-bearing sandstone. The upper sandstone is underlain by a hard gray shale which makes a good floor for shoveling.

The asphalt-bearing sandstones are composed of quartz sand, the grains of which have rough subangular faces which are well pitted and tend to be more or less water repelling because of their affinity for bituminous substances. The grains are cemented more or less by lime carbonate or iron oxide along with the asphalt. The cementing material other than the asphalt is strong enough so that the rock can be crushed to any specified size for coarse aggregate. Muscovite mica is a common constituent of most of the sandstones. It occurs both disseminated throughout the mass and as thin seams, the latter separating the rock into beds varying in thickness from a few inches to several feet.

The texture of the sandstones may vary greatly within the deposits and from one deposit to another, but in any one deposit the average screen analyses will be consistent. The maximum grain size of the quartz sand is usually between 28 and 35 mesh, Tyler screen, and the majority

are found to be about 150 microns in diameter or 100 mesh, Tyler screen. Some typical screen analyses are given in Table 1.

TABLE 1.—*Typical Screen Analyses of Asphalt-bearing Sandstones*

Mesh	1	2	3	4	5	6	7	8
	Percentages							
— ¼ in. + 10	16.0	2	1	1				
— 10 + 20	8.0	3	5	10				
— 14 + 28					Tr.	Tr.		
— 20 + 30	8.0	5	6	4	4.1	3.0	Tr.	
— 28 + 35								2.58
— 30 + 40	7.0	10	5	4	6.8	5.0	6.8	
— 35 + 48								8.55
— 40 + 50	8.0	21	5	5	19.1	11.5	16.3	
— 48 + 65								42.73
— 50 + 80	8.0	35	23	23	41.5	38.0	45.3	
— 65 +100								3.40
— 80 +100	10.0	12	27	26	17.3	24.0	24.5	34.18
—100 +150								
—100 +200	20.0	8	17	17	5.4	9.0	3.8	5.55
—150 +200								
—200	15.0	4	11	10	5.8	9.5	3.3	3.01

1. Reliance Rock Asphalt Co., Ellis, Vernon County.

2, 3, 4. El Dorado Springs, Cedar County.

5. Missouri Asphalt Rock Corporation [Missouri Bur. Geol. and Mines (1926) 19].

6. National Asphalt Refining Co. [Missouri Bur. Geol. and Mines (1926) 19].

7. SE. ¼ SW. ¼ sec. 24-34-30 [Missouri Bur. Geol. and Mines (1926) 19].

8. Sheldon, Vernon County [Missouri Bur. Geol. and Mines (1918) 15].

The color of the asphalt-bearing sandstones varies from a brilliant sparkling black to a chocolate brown on freshly exposed surfaces, and from a dull gray to a dirty brown on weathered surfaces.

Bitumen Content

The bitumen content of the sandstones varies from a trace in the noncommercial portions of the deposit to as high as 14 per cent by weight. The average of the commercial sandstone ranges from 4 to 8 per cent. Usually with the higher percentages of bitumen, especially above 10 per cent, the bitumen contains too many volatile oils which are detrimental to its successful use as road material. In some cases where the bitumen cement is much below 4 per cent, the sand grains are so tightly cemented as to hinder the penetration of asphalt added to bring up the bitumen content. This would make a very unsatisfactory road material.

The beds of sandstones, as defined by the mica seams, may vary greatly in bitumen content, and it is not unusual to find a rich bed overlying a lean one, or vice versa.

The characteristics of the bitumens vary greatly throughout the entire district, and they may vary within an individual deposit. The bitumen may be a light oily fluid resembling kerosene in its characteristics, or it may be a hard, brittle, solid resembling gilsonite. Fortunately these two extremes are the exception rather than the rule. The general run of bitumen is a black, viscous substance with a penetration of 100 to 150; that is, a standard penetrometer needle with a 100-gram weight will penetrate 100 to 150 hundredths of a centimeter in 5 sec. at 25° C. The bitumen contains from 10 to 30 per cent asphaltene and from 90 to 70 per cent petroleum.

THE DEPOSIT

Test pits have shown the deposit to cover practically the entire tract under lease. The thickness of the deposit varies from 6 to 10 ft. of commercial rock with an overburden of 3 to 4 ft. of soil.

This deposit is in the upper sandstone, described earlier, which is thought to be the equivalent of the Red Fork oil sand of Oklahoma. Other deposits at this horizon in territory to the south show a thickness of commercial sandstone up to 15 ft., with 5 to 10 ft. of lean asphalt-bearing sandstone overlying it.

The rock is fine grained and quite uniformly impregnated with bitumen, the content of which varies from 4 to 5 per cent. The gradation of sand grains provides a very stable aggregate. The sand grains have the characteristics described earlier in the paper. The chief cementing material is the bitumen, although some lime carbonate is present. Muscovite mica occurs disseminated throughout the mass, but it is a very minor constituent.

Approximately three million tons is available for quarrying, an amount of material sufficient to surface 3750 miles of 20-ft. highway.

QUARRYING OPERATIONS

The soil overburden is stripped from the rock with a caterpillar tractor and "Fresno type" scraper. So far, the overburden has been placed at one side of the quarrying operations, but as the quarry becomes larger the overburden will be used to backfill the pit. Since the overburden consists of soil only, and the top of the sandstone is smooth and level, this method removes practically all of the overburden and very little subsequent cleaning by hand is necessary before quarrying.

The present quarry face is about 300 ft. long and 9 ft. high, and is carried in one bench. The face is vertical and every effort is made to keep it as uniform as possible.

All drilling is done from the top of the bed with vertical holes drilled 4 ft. back from the face and spaced on 3-ft. centers. Jackhammers with 1-in. hexagon hollow drill steel are used. The rock drills rapidly, one

man with a jackhammer averaging about 200 ft. per day. The bit wears slowly, but the abrasiveness of the rock causes the gage to wear rapidly. Three changes of steel are used, the lengths being 3, 6 and 8 ft. Holes are bottomed about 6 in. above the underlying shale. This gives a clean break between shale and sandstone. The shale is flat-lying, and no "horsebacks" have been encountered to prevent the obtaining of a clean product. Each hole is loaded with six to seven sticks (0.35 lb. each) of Hercomite No. 6 dynamite and 40 to 50 holes are fired simultaneously with an electric blasting machine. Only one row of holes can be shot at a time, advancing the face about 4 ft. If more than one row is shot, the rock breaks back into the face near the surface and causes difficulty in subsequent drilling.

Good shattering of the rock is difficult, because of the extreme resilience of the cementing bitumen. Many boulders broken off by the first shot must be drilled and shot again. All boulders of less than 3 ft. in diameter are broken with a "boulder breaker," similar to a paving breaker. These boulder breakers are made from jackhammers with the rotator mechanism removed, and use 1-in. hexagon steel about 1 ft. long, with a chisel bit. By this method the rock is broken readily along incipient bedding planes. As stated earlier, the sandstone breaks cleanly from the shale floor, and there is no contamination of the product from this source.

The rock is loaded by hand into 2-ton dump trucks (three men loading each truck), four of which normally serve the quarry. It is hauled approximately 1000 ft. to the processing plant, where it is dumped upon an inclined grizzly made of 90-lb. railroad rails spaced with 5-in. clearance between the flanges. The grizzly oversize rolls on to a flat grizzly with the same clearance between rails, where it is sledged through by hand. All undersize falls into a small subsurface hopper from which it is drawn into a 2-ton steel tram car. This car is drawn up an incline by means of a cable and vertical steam hoist and discharges into the 300-ton storage hopper at the processing plant.

PROCESSING PLANT

The essential steps in the treatment are: (1) crushing all rock through a $\frac{1}{4}$ -in. round-hole screen; (2) heating the crushed rock to a temperature ranging from 210° to 250° F.; (3) adding additional asphalt to the hot rock to bring the total bitumen to the required amount; and (4) delivering the finished product into railroad cars or trucks ready for laying on the job.

The capacity of the processing plant is 400 tons of finished product per day. The crushing and mixing equipment is housed in a steel-frame structure covered with corrugated galvanized sheets. The flow sheet is shown in Fig. 1. The storage hopper for the untreated rock is of wooden

construction. The rock is drawn from this hopper and fed by gravity to a No. 5 McCully gyratory crusher which is set to discharge a 1½-in product. This product is fed by a belt conveyor to a set of 36-in. Cornish-type rolls. The primary roll product is elevated by an 18-in. belt and bucket elevator to two 4 by 12-ft. trommel screens. As the

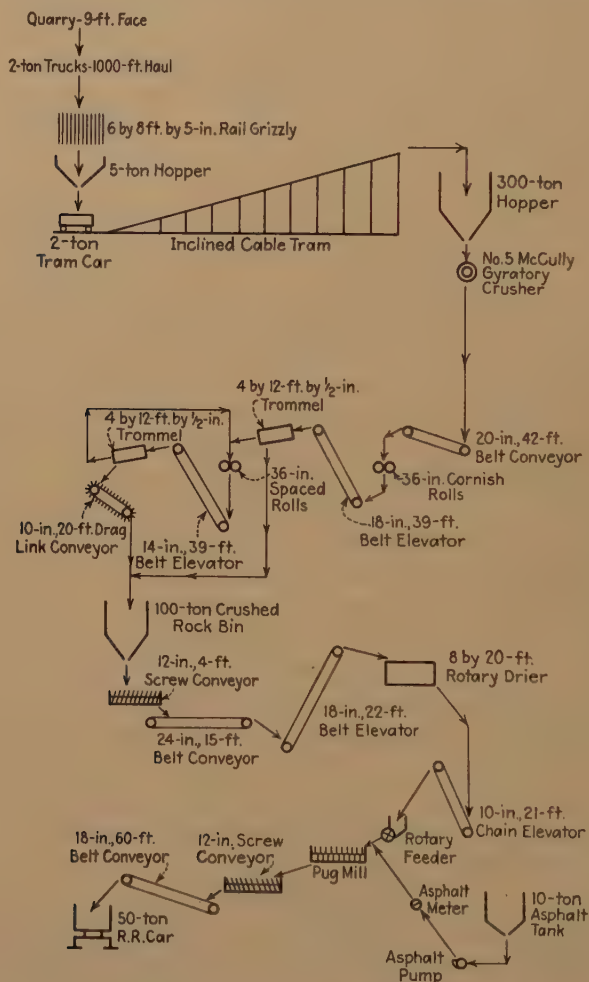


FIG. 1.—FLOW SHEET, RELIANCE ROCK ASPHALT PLANT, ELLIS, MO.

specifications for most of the processed rock asphalt call for all of the aggregate to pass through a 1/4-in. screen, the screen jackets on the trommels have 1/4-in. round holes. The oversize from the two screens is fed by gravity to a set of secondary rolls spaced 36 in. apart and working in closed circuit with the trommel screens. The undersize from the trommel screens is discharged into a 100-ton steel storage bin.

As stated before, the raw rock contains from 4 to 5 per cent bitumen. State specifications for rock-asphalt paving material usually call for either 7 to 9 per cent or 8 to 10 per cent total bitumen. Therefore it is necessary to add from 3 to 4 per cent of refined petroleum asphalt purchased from Mid-Continent refineries. After the rock has been crushed and sized, it is stored in a 100-ton steel bin from which it is delivered into the slowly revolving rotary drier. The drum is 8 ft. in diameter and 20 ft. long and is heated by open flames inside the drum. Fuel oil atomized with steam is used for fuel. The temperature at the discharge end is maintained at about 250° F., a temperature at which all moisture is removed from the rock and the most thorough impregnation of the added asphalt is obtained. Accurate temperature control is obtained by means of a recording thermometer installed in the drier.



FIG. 2.—QUARRY OF RELIANCE ROCK ASPHALT CORPORATION.

The dried and heated rock is elevated to the small surge bin directly over the pug mill and is introduced into the pug mill by a rotary feeder operating at a constant speed, which insures a uniform volume of rock feeding into the pug mill. The asphalt is pumped from the storage tanks at a temperature of about 250° F. through steam-jacketed lines to the pug mill. A Hauck asphalt valve and a recording asphalt meter control the asphalt, which is discharged into the pug mill through a slotted 1½-in. pipe placed directly beneath the rotary feeder that discharges the hot rock. The temperature of the rock and asphalt is maintained throughout the period of mixing by steam coils within the enclosed pug mill. Under ordinary operating conditions the hot rock and asphalt are mixed for five minutes.

The tonnage of hot rock fed to the pug mill and the amount of hot asphalt to be added are under such close control that the bitumen content of the finished product can be maintained within 0.5 per cent for any required tonnage. Determinations of bitumen content of the raw rock and finished product are made in a laboratory at the plant.

Besides the percentage of bitumen present, the type of asphalt is important. If the asphalt is too "soft," it is unsuitable for most temperate climates, but if it is too hard it will crack and spall during cold weather. Normal specifications call for asphalt of 85 to 100 penetration for material to be used in the spring or fall of the year. For material to be used in summer, a slightly lower penetration is recommended.

The processing machinery is driven from a line shaft by a 120-hp. Fairbanks-Morse oil engine. (A friction clutch is provided for starting the engine.) Compressed air for drilling at the quarry and for miscellaneous purposes around the plant is provided by a Sullivan air compressor, belt-driven by a 100-hp. Fairbanks-Morse oil engine. Electricity for lighting is supplied by a 6-kw. direct-current generator.

Water for cooling the oil engines, for boiler use and for drinking purposes is obtained from a 500-ft. drilled well, equipped with an air-lift, and comes from a break in the Boone limestone. It is suitable for use in the boilers without treatment.

Steam for hoisting the raw rock into the storage hopper, for atomizing the fuel oil in the drier, and for heating the asphalt is generated in a 150-hp. horizontal boiler fired by fuel oil. The hot asphalt is stored in two 10,000-gal. tanks buried in the ground and provided with steam coils for heating. The asphalt is obtained from the refineries in insulated tank cars and is discharged into the underground storage tanks while still hot.

Laboratory Control

A laboratory at the plant insures a finished product that complies with the specifications. Bitumen determinations requiring extreme accuracy are made by the "rotarex" or centrifuge method, the bitumen being dissolved in carbon disulfide and extracted. For ordinary plant control bitumen determinations are made by ignition, and a factor of 1.5 per cent bitumen is deducted from the results to give the true bitumen content. This difference between ignition and extraction determinations is due to moisture contained in the rock and a small amount of lime which occurs as a cementing material, part of which is dehydrated.

The penetration of the asphalt is determined by means of a penetrometer.

To determine the stability of the finished product, small briquets about 3 in. in diameter by 1 in. thick are made up. A hydraulic press, using about 1000 lb. pressure per square inch, is used to compress the aggregate. These briquets are allowed to age for three days and then are tested for cohesion and ability to withstand compression.

The maximum size of the particles in the aggregate is usually $\frac{1}{4}$ in., although aggregates have been laid on test roads with particles ranging up

to $1\frac{1}{2}$ in. For most work the $\frac{1}{4}$ -in. maximum has been found to give the most satisfactory service.

ACKNOWLEDGMENTS

The authors wish to acknowledge the cooperation of the several producers in the area discussed, and to thank Mr. C. E. Heinz, consulting metallurgist of Joplin, Mo., for his assistance in gathering together the data incorporated herein and for his helpful criticism of the completed paper, and Mr. M. D. Harbaugh, Secretary of the Tri-State Zinc and Lead Ore Producers Association, for his suggestions and criticism.

DISCUSSION

J. R. THOENEN,* Washington, D. C.—Is there any trouble with material sticking to the belt? In Kentucky, such trouble occurs to some extent.

W. M. WEIGEL,† St. Louis, Mo.—Some rock containing the lighter asphalts would be troublesome, but the quarrymen select the drier, free-of-oils rock.

* U. S. Bureau of Mines.

† Mineral Technologist, Missouri Pacific Railroad Co.

The Wire Saw as a Tool for Cutting Slate and Building Stone

BY OLIVER BOWLES,* MEMBER A.I.M.E.

(Pennsylvania State College Meeting, September, 1936)

WHEN a new type of equipment revolutionizes methods of quarrying one kind of stone, producers of other kinds focus their attention on its potentialities in their particular fields. The purpose of this paper is to discuss the accomplishments of the wire saw in slate, to show its more limited attainments in other kinds of stone, and to point out the possibilities of much wider application.

A wire saw consists of a $\frac{3}{16}$ -in. or $\frac{1}{4}$ -in. three-strand wire cable that runs as a belt and cuts the rock by abrasion. (Fig. 1.) The teeth of the saw are sand grains carried in the grooves between the strands.

ACCOMPLISHMENTS IN SLATE†

Initial experiments with wire saws by the U. S. Bureau of Mines in the Colonial Slate Quarry, Wind Gap, Pa., in 1926, attained success so immediate and pronounced that within two years in virtually every quarry in the neighborhood of Wind Gap, Pen Argyl and Bangor, the most productive slate area in the United States, wire saws were substituted for channeling machines. An 80-ft. cut in slate is shown in Fig. 2. The technique of operation has been described in several reports (references at the end of the paper). The present purpose is to summarize the accomplishments in a large, typical slate quarry after nine years of experience.

Parsons Bros. Slate Co., at Pen Argyl, Pa., substituted two wire saws for six channeling machines in 1927. The following comparisons are made between the present method with wire saws and the former channeling-machine method.

Rate of Sawing.—The average rate of cutting with a wire saw in the early days of its operation, as determined by the Bureau of Mines¹, was 14 to 17 sq. ft. per hour. Parsons Bros. Slate Co. estimates the present rate of cutting at about 25 sq. ft. an hour, whereas the channeling-machine

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† Data for accomplishments in slate were supplied through the courtesy of Martin L. Parsons, of Parsons Bros. Slate Co., Pen Argyl, Pa.

rate was about 4 sq. ft. Thus the wire saw cuts about six times as fast as the channeling machine.

Equipment Cost.—The six channeling machines originally used in this quarry represented an outlay of about \$18,000. The wire-saw equipment substituted therefor, including a core-drilling machine, eight standards or posts with their attachments, and two motor drives, cost about \$4250,



FIG. 1.—WIRE AT POINT WHERE IT LEAVES CUT. FIG. 2.—WIRE SAW MAKING CUT 80 FEET LONG IN SLATE.

or less than one-fourth of the investment required by the former method. Expendable equipment in the form of channeling-machine steel costs a great deal more than the wire that takes its place. Incidentally, a general trend toward wire saws throughout the dimension-stone industries would have an appreciable influence on the iron and steel trade because it involves a declining demand for bar steel and a marked increase in the requirements of steel wire.

Labor Cost.—Two men only are needed to operate the two wire saws, which are so automatic that the men can at the same time operate the core drill. Each of the six channeling machines originally used required two men and demanded the services of more highly specialized mechanics. It is estimated that the labor cost for wire-sawing per square foot of

surface obtained is only about 10 per cent of that required for channeling. Also, when channeling machines were used a blacksmith and a helper were on constant duty, and they are not needed at all for wire-saw work.

Power Saving.—To operate six channeling machines, together with drills and other pneumatic equipment, a large air compressor handling 1710 ft. of air per minute was required. It was driven by a 300-hp. motor, and the initial cost was about \$15,000. When wire saws were substituted the use of this equipment was discontinued, for it was found that a small compressor of 400 cu. ft. capacity requiring a 75-hp. motor and costing only about \$1000 was ample for all requirements. The wire saw uses no compressed air. It is driven by a 10-hp. motor and the core drill by a 20-hp. motor, consequently the electric power demands are small.

Waste Elimination.—Waste has always been a serious problem with slate producers. Estimates made before wire saws were used indicated that 70 to 95 per cent of the gross production was thrown away. This condition was in part the incentive that led to the development of better equipment. With these saws, the quarry waste is only one-third the earlier amount. Obviously quarry production has increased greatly without any increase in gross output.

RESULTS ATTAINED IN OTHER ROCKS

Limestone.—Newsom² described wire-saw operations in a quarry operated by the Bloomington Limestone Co. of Bloomington, Ind. He calculated a rate of 52 sq. ft. per hour actual cutting time in solid oolitic limestone. With due allowance for moving and other interruptions, an average rate of about 40 sq. ft. per hour was estimated. An average rate of channeling determined on the basis of actual time for which channel runners are paid ranges from 10 to 15 sq. ft. per hour. Newsom estimated that the cost of cutting with a wire was about one-half that of channeling and that there was a substantial reduction in the proportion of waste. Newsom proved also³ that a wire could be used successfully for making horizontal cuts in limestone.

The Monan Stone Co. has used wire saws in its quarry near Bloomington, Ind. When observed by the writer in 1931, the wire was attaining a rate of 26 sq. ft. per hour, cutting unusually hard surface rock. No cuts had been made on commercial stone.

Marble.—Many years ago the Colorado Yule Marble Co., of Marble, Colo., used a wire saw to cut a mass of marble lying between two quarry openings. This was probably the first use of such equipment in an American quarry. No data are available as to rate or cost of sawing, but the results were satisfactory. Weigel⁴ described the operation of a wire saw in a marble quarry in Arkansas in 1930. For one typical cut a rate of 10.7 sq. ft. per hour was attained, and the direct cost was less than 10¢ a square foot of surface sawed.

The Vermont Marble Co. experimented with a wire saw at West Rutland, Vt., in 1927 and 1928, but found that the cutting rate was too slow and the wear on the wire too great to justify its use. Further tests were made in a more favorable quarry in 1932 and 1933. According to Bain⁵, a cutting rate of 5.31 sq. ft. per hour was attained, actual running time, or 2.67 sq. ft. per hour when time for setting up equipment and all other interruptions are included. The direct cost of cutting with a wire was 48¢ a square foot as against 42¢ for the channeling machine. For deep cuts, however, the advantage lies with the wire. Marble abrades the wire rapidly. An average of over 4 ft. of wire was required for each square foot of surface cut whereas in limestone and slate only about one foot of wire is needed for each square foot cut.

Sandstone.—According to report, wire saws were used successfully in a sandstone quarry near Mitchell, Ind., in 1930 and 1931, but no reliable data are available. The Cleveland Quarries Co. made several tests with a wire saw in a quarry at Amherst, Ohio, in 1935, but has not yet submitted detailed figures.

SAWING IN YARD AND MILL

Wire saws have been used for scabbling (trimming) marble and limestone quarry blocks. Several blocks may be lined up and trimmed at the same time with a single wire. A Tennessee marble company records a rate of 6 sq. ft. per hour when two blocks are cut simultaneously.

The Taylor Stone Co., McDermott, Ohio, has used wire saws for many years for jointing blocks of sandstone; that is, for cutting them off in specified lengths. Many blocks or slabs are placed on a car and cut simultaneously. The wire is fed downward by worm gears at a rate of 24 in. an hour.

During recent years, wires have been used in stone-finishing mills for roughing-out blocks to approximately the shape desired, and even for cutting intricate designs. Grill work is made by guiding the wire with a metal template or pattern. Statuary is shaped with wires even to the extent of outlining facial features, thereby relieving the stone cutter of much laborious hand cutting. An ingenious method has been devised for cutting columns ready for smoothing and polishing. The wire runs lengthwise with the rectangular block, which is mounted on cars. The necessary curvature of the cut is attained by moving the block slowly in a lateral direction while the wire descends.

ACCOMPLISHMENTS IN EUROPE⁶

About 150 wire saws are used in the famous Carrara marble district, and about 20 in the travertine quarries at Tivoli, near Rome. Wire saws are used extensively in the marble quarries and mills of southern France. About 80 per cent of the 42 marble and limestone quarries of Belgium use wire saws. According to report, an average rate of sawing Italian

travertine is 5.4 sq. ft. per hour, and the cost is about 20¢ per square foot. Rates of cutting various calcareous rocks in Belgium range from 4.3 to 5.4 sq. ft. per hour. The cost of sawing in the quarries is said to be about 51¢ a square foot, while sawing detached blocks costs about 29¢ a square foot. Limestone, some of which is crystalline enough to be classed as marble, quarried at Soignies near Brussels, saws at a rate of 7 to 9 sq. ft. an hour. Wire saws have been used successfully in these quarries for a great many years.

POSSIBLE SIMPLIFICATION OF EQUIPMENT

In the slate district of Pennsylvania, the equipment has been improved very greatly since the first experiments were made, and a consequent gain has been reflected in efficiency of operation. The results relating to American limestone and marble quarries in preceding paragraphs of this paper for the most part represent rather crude attempts by inexperienced men with untried equipment. It may be assumed that continued study and experimentation will result in great improvements both in the machinery used and in the technique of operation. As an illustration of possible improvements, the writer observed in a quarry in Belgium in 1935 a very simple type of core drill gear-driven from a sheave rotated by a wire belt. Its cost would be only a fraction of that required for the standard core drills now used in America.

CAN WIRE SAWS BE USED MORE EXTENSIVELY?

Stone is an extremely hard, resistant material. The process of separating primary blocks is probably the most costly of all processes in quarry operation, and therefore the one to which quarrymen should devote their most earnest efforts toward improvement. Stone is meeting with increasing competition from a multitude of synthetic products, some of which are of high quality and most of which can be manufactured by mass-production methods. To maintain its position or win wider use in the competitive field, the stone industry must devote increasing attention to economies in production and must utilize to the fullest extent every new device or method that will apply.

The fact that wire saws have been used successfully for many years in cutting marble, limestone and travertine in European quarries, where labor costs are generally lower than in America and where mechanization has been promoted less vigorously, implies backwardness and lack of aggressive experimentation on the part of American quarry operators. For most quarry operations, American equipment is superior to that used in Europe, but in making primary cuts, the most important of the processes, American quarrymen, with the exception of those in the Pennsylvania slate and Vermont marble areas, lag behind. Until this condition is improved stone producers will suffer a handicap in the competitive field.

The introduction of the new equipment in the Pennsylvania slate district met at first with much opposition and skepticism, but the advantages gained surpassed even the fondest expectations of those who were most optimistic. However, before success was attained many difficult problems were encountered and solved. Similarly, marble, limestone and sandstone quarrymen may meet with seemingly insurmountable obstacles, but the goal is promising enough to justify the most persistent efforts.

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DISCUSSION

MEMBER.—Will you tell us something about the use of the wire saw for cutting footings for dams?

O. BOWLES.—In Maryland, the wire saw was used to cut a shaly or schistose rock so as to get a solid foundation for a dam. At this location, blasting would have shattered the rock and weakened it, but the wire saw, since it cuts by abrasion, causes no shattering. Another interesting application outside of the dimension-stone industry may be in the making of subway cuts. It is also used for facing rock cuts for highways. An unusual application of the wire saw has already been described by J. B. Newsom² who employed it to put in a large horizontal cut in an Indiana limestone quarry. At many Indiana limestone operations there is a shale parting, which serves to cushion the shock when the overlying rock is blasted, but at this particular operation this shale parting was absent. The purpose of the wire-saw cut was to separate the overlying inferior rock from the good stone. When making the cut with a wire saw, the inferior rock could be removed by blasting without damaging the good stone below. Otherwise, it would have been necessary to remove the overlying rock by channeling machines at very largely increased cost. This work proved that horizontal wire-saw cuts are possible. It is significant that the wire saw is more widely used abroad, especially in marble quarries, despite the fact that the European operations are not so highly mechanized as our own in other respects. The cost of wire-saw cutting is remarkably low, and the wear of the wire is not excessive. A thousand feet of wire will cut about 1000 sq. ft. of surface and costs, on this basis, less than $\frac{3}{4}$ ¢ per foot. I confidently expect that this method of making primary cuts will be adopted eventually for quarrying all of the softer kinds of stones.

MEMBER.—In Switzerland wires or ropes are commonly employed for cutting glaciers, clearing away snow masses and avalanches.

MEMBER.—The same principle is employed, I understand, for delicate surgical operations on the skull.

A Limestone Mine in the Birmingham District

By C. E. ABBOTT,* MEMBER A.I.M.E.

(New York Meeting, February, 1935)

THE Birmingham district, Alabama, is distinctive in the proximity to one another of its deposits of iron ore, coal and flux. These three basic requisites for the making of iron and steel are found within a radius of 10 miles. Both dolomite and limestone are produced, the former for blast-furnace flux and the latter for basic open-hearth operations.

The Tennessee Coal, Iron and Railroad Co., a subsidiary of the United States Steel Corporation, operates a limestone mine at Muscoda, near Bessemer, 12 miles southwest of Birmingham, served by its own railroad transportation system. The mine has a capacity of 130 tons per hour. Average analyses of a recent month's production are as follows:

	SiO ₂	$\frac{\text{Fe}_2\text{O}_3}{\text{Al}_2\text{O}_3}$	CaCO ₃	MgCO ₃
Lump, per cent.	0.51	0.22	98.63	0.14
Crushed and washed, per cent.	0.70	0.26	98.36	0.18

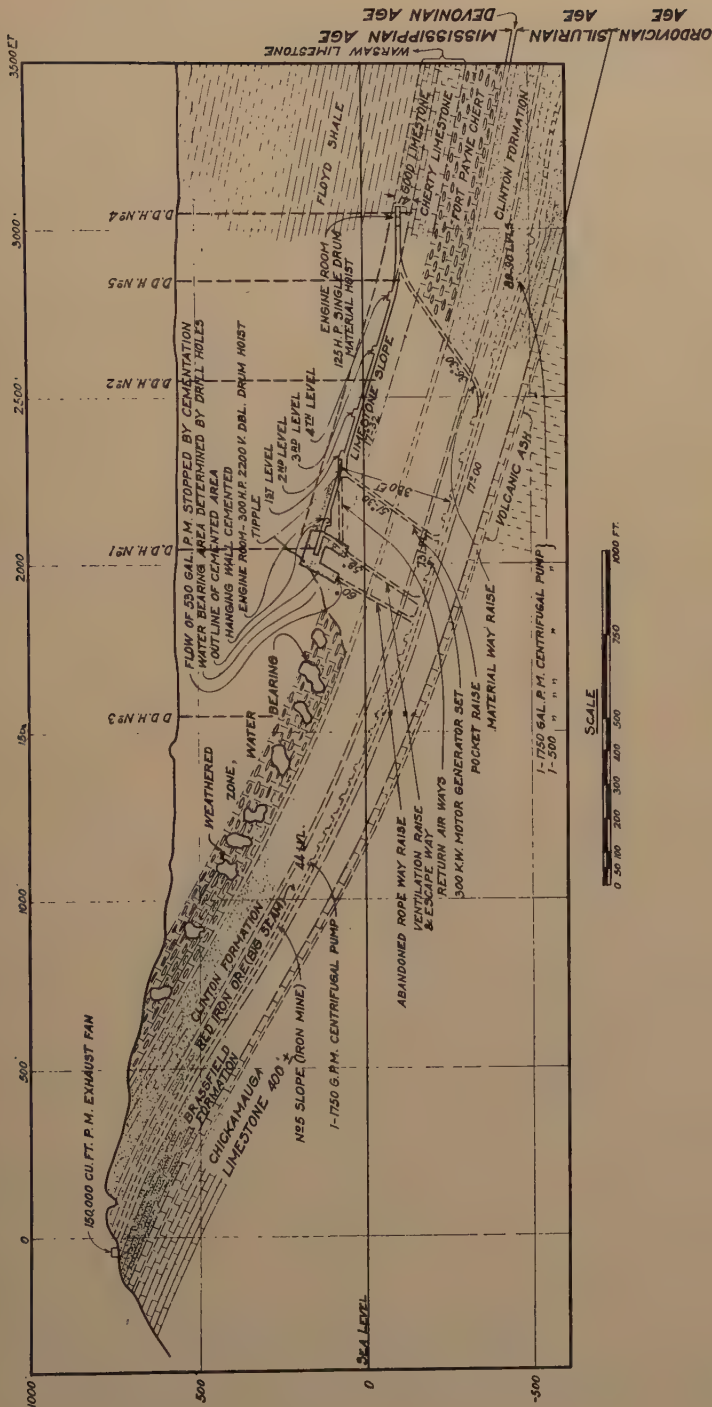
Lump stone, 2½ to 7½ in., is used in open-hearth operations at Fairfield Steel Works. Fine crushed and washed stone, 1¼ to 0.063 in., is calcined at the lime-burning plant and used for lime charges at the Ensley open hearths.

GEOLOGY

The district's deposits of hematite ore occur in the Clinton formation, and the limestone beds mined are stratigraphically 330 ft. above the ore beds worked. A cross section on the center line of the mine is shown in Fig. 1. This limestone formation, 130 ft. thick, is in the Warsaw horizon of the Lower Mississippian. It lies above 100 ft. of Fort Payne chert, which is immediately above the Clinton formation, and is composed of layers of chert and cherty limestone, approximately 50 per cent of the material being pure chert. The chert is extremely hard, and the cost of driving raises through it is very high. Both iron ore and limestone deposits lie on the southwest limb of the Jones Valley anticline, the formations dipping from 15° to 32° to the southeast. The iron-ore seam outcrops along the crest of Red Mountain, a ridge with northeast-southwest trend rising 200 to 300 ft. above Jones and Shades valleys at the

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* Vice President, Tennessee Coal, Iron & Railroad Co., Birmingham, Ala.



base. The limestone bed has been badly weathered for approximately 1200 ft. down the dip from the outcrop, and this portion is not workable.

EXPLORATION

In 1909 considerable diamond-drilling was done to determine the extent and composition of the iron-ore seam, and to secure information as to structural conditions at greater depth. These drill holes indicated the presence of a bed of limestone 330 ft. above the Big (iron ore) seam. Further investigation of the cores in 1925 showed a lower silica and higher lime content in the vicinity of Muscoda than at any other location on Red Mountain, and also higher in grade than in the surface deposit near Chepultepec. In 1926, seventeen diamond-drill holes were put down from surface at Muscoda iron mines Nos. 4, 5 and 6. The location of the holes was selected to prove the extent and grade of the deposit, and also to locate the lower limit of the zone of weathering in order that future developments could be confined to unweathered stone. This was necessary because the weathered limestone and chert in the Warsaw and Fort Payne formations are heavily water-bearing, as indicated by inflows of over 3000 gal. per minute for considerable periods at iron mines where the ore pillars have been extracted in territory underlying the zone of weathering. The drilling indicated the presence of a deposit of high-grade limestone from 40 to 50 ft. thick.

DEVELOPMENT

It was decided to use the temporarily abandoned slope of iron mine No. 5 for development of the limestone mine (Fig. 2). An 8 by 12-ft. incline for material and manway was raised from the ore slope to reach the lower edge of the limestone deposit. From the top of the incline a 10 by 20-ft. slope was raised a distance of 900 ft. in the base of the merchantable limestone. Concurrently, two additional raises were started from the ore slope, the upper for a ropeway from the limestone hoist, which was to be located in a level in the ore seam, and the lower 60 ft. down the ore slope, to be utilized as a transfer pocket between limestone and ore slopes. The rope raise was located below the known zone of weathering, as indicated by the diamond-drill cores, and was raised in unweathered material to a height of 259 ft., or to a point 83 ft. below the limestone, where drilling encountered weathered limestone and chert and resulted in a flow of 530 gal. per min., which made it advisable to stop the work. Work in the pocket raise was also discontinued at that time. Cementation was then attempted through a series of long holes drilled with pneumatic drills and sectional steel, and diamond-drill holes radiating from the upper portion of the limestone slope. By means of pressure pumps, 27,000 sacks of cement were injected into this area and the inflow of water from the rope raise reduced

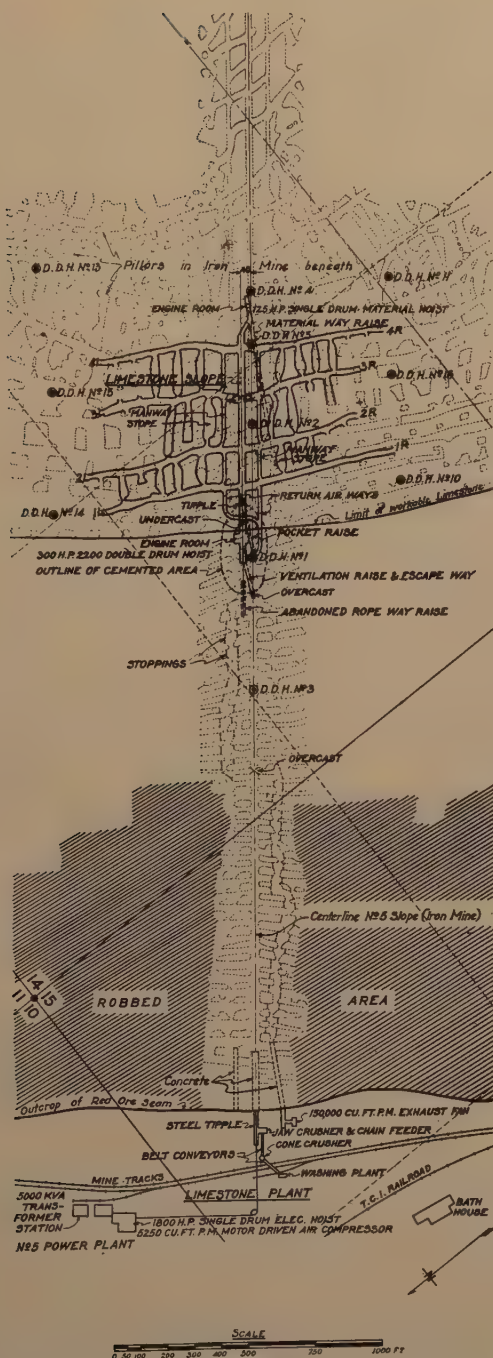


FIG. 2.—PLAN OF LIMESTONE MINE AND OLD IRON MINE.

to 20 gal. per min. It was found, however, that the territory ahead of these workings contained many openings filled with mud and clay that could not be replaced by cement, making it unsafe to continue work in the rope raise and slope because of the probable inflow of large quantities of mud and water. A 12-ft. plug of broken rock was placed in the upper end of the rope raise, injected with cement, and the raise abandoned. Additional cementation work was done in the back and sides of the upper end of the stone slope, where an inflow of approximately 250 gal. per min. was substantially reduced. Plans were altered and the site for the auxiliary hoist for handling stone was changed to the upper end of the limestone slope. The proposed pocket raise was completed and utilized for ventilation and escapeway purposes, and a new pocket raise started from a point 150 ft. down the ore slope. In the meantime four pairs of 12 by 20-ft. working levels were being driven from the stone slope. These were turned at right angles from the slope at 200-ft. intervals and driven on a grade of 1.5 per cent in favor of the loads. Mechanical leaders were used for driving the levels and raising the manways.

MINING

Drilling and Blasting.—Standardization of rock drills, steel, and methods has been accomplished in the mines of this company. The drills used for drifting and stoping are 4-in. wet-type hammer drills; for plug drilling and miscellaneous purposes, the jackhammer type is used. A few dry stopers are used for drilling in short raises at the upper end of each stope. Heavy drills use $1\frac{1}{4}$ -in. hollow-round steel, and the jackhammers $\frac{7}{8}$ -in. hexagon. The double-taper cross bit with 7° and 14° angles is standard for all starters. The Carr chisel bit is used on the longer steels. All 4-in. hammer water drills are mounted on tripods. A center cut round is used in driving levels, stopes and cross-cuts, 10 ft. being the maximum depth of hole. Pilot holes 20 ft. long are drilled ahead of working faces to give advance notice of water-bearing joints or openings. For tamping, fine limestone cuttings packed in paper tamping bags are used. The practice is to use three 18-in. bags of tamping in each hole. The explosive is a water-resisting, ammonia dynamite with a bulk strength of approximately 42 per cent. No. 6 caps and countered fuse are used in all blasting except narrow development work and sinking operations, where electric detonators are used. Blasting is done only at the end of each shift. Blasting operations are planned so as to produce as much lump stone as practicable.

Stoping.—Stope openings 12 by 12 ft. are broken off the mining levels on 70-ft. centers. Loading ramps are constructed at each stope opening for mechanically operated scraper loading. Steel channels are used for construction of a scraper loading platform extending from the footwall

to a height sufficient for stone from scrapers to fall into tram cars. After a stope has advanced 20 ft. from the upper rib it is widened to 30 ft. and driven to within 15 ft. of the lower rib of the level above, then it is narrowed to 8 ft. for the remaining distance and broken through near the hanging wall to properly ventilate the stope. There remains 25 to 30 ft. of good limestone in the back of the stope, which is mined by starting back stoping at the lower end of the stope until the lower portion of the permanent hanging wall is reached, leaving sufficient broken stone to work on. Back stoping is then conducted in successive cuts each about 8 feet thick from bottom to top, and continuing from the lower end to the upper. Drilling is done at approximately right angles to the dip.



FIG. 3.—SCRAPER LOADING FROM STOPE TO MINE CAR.

Drilling and quarrying is done on top of broken stone left in the stope, so that very little wooden staging is required. Enough stone is removed from the stope to provide working space at the back; all broken stone is removed after drilling and blasting are completed. When the next stope toward the face of the level is broken into the level above, a stopping and adjustable air regulator is installed near the hanging wall at the upper end of the worked-out stope, so that the ventilating current is diverted toward the face and any gas being made in the stope will also be removed by the air current. Practically no timbering is required in the levels and none in the stopes.

Mechanical Loaders and Scrapers.—All stone in levels and stopes is loaded by eight mechanical loaders (Fig. 3) into $3\frac{3}{4}$ -ton standard-gage, roller-bearing tramcars. Each mechanical loader consists of a double-drum self-propelling hoist on standard-gage trucks driven by a 50-hp., 250-volt, d.c. motor, equipped with contactor controls. Loading is done by means of a 10-in. skidder block at the working face and an in-haul

and out-haul wire rope handling a scraper of $1\frac{1}{2}$ cu. yd. capacity. The scraper is 54 in. wide, 54 in. long and 34 in. high, with teeth in the back end; it is bottomless and weighs 2550 lb. The speed of the scraper is approximately 150 ft. per min. This type of loader is standard equipment in all iron mines of the company, there being a total of 96 units in operation. Power for mechanical loading in the limestone mine is furnished by an automatically controlled 300-kw. motor-generator set located in the ore slope below.

Tramming and Haulage.—Cars are trammed by gravity from the loading ramps to the limestone slope; empties being returned by mules.



FIG. 4.—SKIP IN ORE SLOPE BEING LOADED FROM MEASURING BIN.

As the length of haul increases, electric trolley locomotives will be installed. Tracks are standard gage of 60-lb. rail, laid on creosoted cross ties, 7 by 9 in. by 8 ft., and the rails are bonded.

Dumping and Hoisting.—Tramcars from the limestone levels are dumped into $3\frac{3}{4}$ -ton balanced skips, which are handled by a 300-hp. a.c. double-drum hoist. This hoist is located at the upper end of the limestone slope, and is equipped with magnetic control, hydraulic-control post brakes and clutches; Lilly hoist controllers with track-limit switches at the tippie, and with a visible skip-position indicator. The skips dump over a double-track steel tippie into a pocket connecting to the ore slope below, where a steel air-controlled undercut gate regulates the feeding of stone into a steel measuring hopper holding 10 tons. An air-operated tripping device allows stone to discharge quickly into the main skip (Fig. 4). The gates of the measuring pocket are then closed by counterweights. The standard-gage, roller-bearing skip, of 10 tons capacity, has three-point suspension on the trucks, and makes a round trip to the surface, 2400 ft. away, in three minutes.

Pumping.—Water from the mine drains to two sumps in the ore slope below, and is pumped to the surface by two motor-driven centrifugal pumps handling 1750 gal. per min., operating in relay. A motor-driven centrifugal pump handling 500 gal. per min. is used in an emergency to send the water to a pumping station in an adjoining iron mine.

Ventilation.—During the development of the mine, ventilation was satisfactorily maintained by means of a booster fan of 50,000 cu. ft. per minute capacity, installed in the ore slope near the foot of the material way to the mine. After the mine had been in operation four months, a large feeder of methane gas was encountered in the 2d right level. This



FIG. 5.—VENTILATING FAN AND BUILDINGS THAT HOUSE DRIVE, SHOWING AIR DUCT AND EXPLOSION DOORS.

feeder and a small one in the 4th right level were making approximately 250 cu. ft. of methane per minute, which, with the gas made by the remaining workings, was sufficient to produce an explosive mixture in the entire mine, should there be a power interruption of 2 hr. duration. The gas was associated with a light oil, which probably seeped into the mine through joint planes, and undoubtedly came from an overlying bituminous shale. Additional ventilation was imperative and an exhaust fan furnishing 150,000 cu. ft. per minute was installed at the surface near the entrance to the ore slope. Installation at the surface was necessary to prevent recirculation of air. The ore slope was used as an intake; the ventilation current traveling down to the material-way up to the limestone slope, thence through mine workings and air courses to the ventilation raise, thence through an overcast on the ore slope into the return airway paralleling the ore slope, and thence to the fan at the

surface. Doors were placed at each level and the ventilation current was divided into two splits, one for each side of the mine. The limestone mine requires 25,000 cu. ft. per minute, the balance being used for ventilation of adjoining ore mines Nos. 4 and 6. To properly ventilate working faces, a blowing fan (2800 cu. ft. per minute) with flexible canvas tubing was placed in each level to maintain ventilation between the last crosscut and the face. A certified gas inspector makes a complete examination of all working places in the mine before each shift goes on. The main ventilating fan is driven by a 125-hp. 2200-volt motor and has an emergency drive consisting of an 80-hp. gasoline engine. The fan is driven through friction clutches and multiple V-belts. In case of an electric-power interruption and slowing down of the fan, an automatic signal is sounded and a blinker system, which is constantly in operation, goes out, warning the attendant, so that auxiliary fan drive may be started.

Compressed Air.—Compressed air for drilling and other purposes is furnished by electrically driven, two-stage air compressors; pressure used is 85 lb. gage at the compressors. Air for the limestone mine and iron mines at Muscoda is supplied by three motor-driven compressors, one furnishing 5250 cu. ft. per minute and two 4000 cu. ft. per minute, and all equipped with air filters.

Lights, Telephones and Signal System.—Both the limestone slope and the ore slope are adequately illuminated. Incandescent lamps of 200 watts are suspended from brackets placed on 60-ft. centers throughout the entire length of these two slopes. A flood light at the head of the man and material-way raise furnishes its illumination. Current for these lamps is supplied from the 220-volt a.c. circuit. In the mining levels, electric lights are tapped on to the 250-volt d.c. mechanical-loader circuit. In addition to these drops, flood lights are located at the loading ramps.

Magneto-type telephones are used for communication and are located at all vital points both on the surface and underground, resulting in almost immediate contact with any phase of the operation.

The signal system employed in the ore slope and the man and material-way where men are lowered and hoisted is rather unique. It consists of a $\frac{5}{16}$ -in. bare messenger wire hung the entire length of these slopes at a height of 6 ft. above the top of the rail and 18 in. from the side of the skip. A special rod is so attached to either end of the skip by means of a flexible shunt wire that by touching this rod to the messenger wire at any point on the slope the electrical circuit is completed through the rail and hoisting rope that acts as a ground. This circuit is 22 volts and actuates a 110-volt relay in the engine room.

The signal system in the limestone slope where only stone is hoisted consists of a two-conductor lead-encased cable in conduit lying on the

crusher set to produce a $7\frac{1}{2}$ -in. product. Crushed stone is retarded by a chain drag to decrease the impact as it is chuted from the crusher to a 36-in. incline belt conveyor, which transfers the stone to a 4-ft. by 8 ft. 6-in. vibrating scalping screen with $2\frac{1}{2}$ -in. square openings. A 30-in. diameter magnetic head pulley on the conveyor recovers tramp iron before discharge to screen. Stone over the $2\frac{1}{2}$ -in. screen is dumped through a chute direct into railroad cars on a siding, for shipment to the open-hearth plant. An air-operated gate regulates the flow of stone and permits the spotting of cars without delay to the plant. Stone under $2\frac{1}{2}$ in. from the scalping screen gravitates to a $5\frac{1}{2}$ -ft.



FIG. 7.—SURFACE TIPPLE AND PART OF CRUSHING PLANT.

cone crusher with $1\frac{1}{4}$ -in. opening. After crushing, it is chuted to a 24-in. belt conveyor and transported to the top of the washing plant, where it discharges to a 4-ft. by 8 ft. 6-in. double-deck vibrating and washing screen with $1\frac{1}{4}$ and $\frac{3}{4}$ -in. openings, respectively, on the top and bottom deck. Three sizes of stone are produced, which are all fed to another similar screen with $\frac{1}{4}$ and 0.063-in. openings. This method of screening is intended to remove the fines and thoroughly wash the stone, removing all dust by means of pressure water sprays on the screens. All sizes are chuted together into railroad cars, a flap gate controlling the flow through two chutes so that empty and loaded cars may be handled without spillage of stone and delay to the plant. This stone is shipped to the company's lime-burning plant to be calcined for open-hearth use. All fine material passing through 0.063-in. screen on the second washing screen is discharged into a conical steel sump

tank with all wash water, from which it is handled by a 4-in. sand pump to a 90,000-gal. conical steel settling tank. The tailings from the settling tank flow into a waste pond and the clear water overflow returns by gravity to the washing system. A concrete drip pad under the washing plant collects drippage from the plant and cars, which is automatically pumped back into the settling tank and water-recovery system. The plant is all steel and concrete construction, and is enclosed with copper-bearing corrugated steel siding and roofing.

SURFACE HOISTING PLANT

The main hoist has a single drum keyed to the shaft; the drum being 10 ft. in diameter with a 13-ft. face. It is driven through single-reduc-



FIG. 8.—SKIP DUMPING AT SURFACE TIPPIL.

tion herringbone gears by an 1800-hp. a.c. motor. The hoisting speed is 2000 ft. per minute. The hoist is housed with a 5250 cu. ft. per minute compressor and has double post hydraulically controlled brake and also a dynamic braking system. It is equipped with Lilly automatic hoist controller and track-limit switches to prevent pulling the skip over the tippel. In addition, a visual skip-position indicator, electrically controlled, shows successive positions of the main skip at various points while dumping at the surface tippel, and enables the hoisting engineer to spot the skip accurately. The hoisting cable is $1\frac{3}{8}$ -in. in diameter and runs from the hoist to a 10-ft. dia. turn sheave at ground level, thence to a 10-ft. sheave at the top of the stone tippel. Roller-bearing idler sheaves are used both on surface and underground.

POWER

Electric power used at this plant is produced by steam turbine-driven generator equipment of the company at Fairfield Steel Works, where blast-furnace gas, supplemented by powdered coal, is used for fuel. Power is transmitted at 44,000 volts to the substation at the mines, where it is reduced to 2200 volts for general use on large motor units, both surface and underground. Auxiliary fans in the levels use 220 volts a.c. current, which is used also on surface vibrating screens and belt conveyors. A small motor-generator set furnishes direct current for the magnetic pulley, and for variable-speed motor operating the chain feeder at the primary crusher. The high-voltage motors are mostly across-the-line starting type.

WAGE SYSTEM

All men in the mine are paid on the basis of hourly occupational rates, the working shift being 8 hr. No bonus is paid and no tonnage rates, and the foremen are directly responsible for the supervision of the work. All tools and blasting supplies are furnished by the company. Miners' lamps and carbide are bought by the individuals using them.

CONCLUSION

It has been the aim of this paper to cover, in a general way, the important features connected with the development of the mine and of underground and surface construction. The encountering of a considerable quantity of methane in this mine was unexpected, and was so serious as to require prompt measures to insure safety to men and property. The weathered condition of the formations at the upper margin of the workable deposit required a change in the plans for development, after it was found that cementation would not eliminate the water hazard.

Mining operations have been conducted since May 13, 1931, the only interruption being the time required for the installation of the main ventilation system.

ACKNOWLEDGMENTS

The plant was designed by the Engineering Department, and was erected and equipment installed by the operating and construction forces of the company. This work was under the supervision of Mr. E. M. Ball, General Superintendent, Iron Mines and Quarries; Mr. R. Faull, Mechanical Engineer, Mining Department; Mr. A. W. Beck, Chief Engineer, Iron Mines and Quarries; and Mr. Ewing Carter, Superintendent, Muscoda Division.

The photographs are reproduced by courtesy of *Pit & Quarry*, from the issue of Oct. 5, 1932.

DISCUSSION

J. R. CUDWORTH,* University, Ala. (written discussion).—Mr. Abbott's paper describes a unique mining enterprise for the Birmingham district, which previously had not attempted any underground mining of fluxing materials. The value of saving diamond-drill cores for future reference in connection with possibilities not even considered at the time of drilling is well brought out.

One of the unusual features of the deposit was the presence of methane gas in considerable quantities in this Lower Mississippian limestone. The prompt and efficient installation of an artificial ventilating system for the whole mine was required before safe operation was possible. Of particular note was the use of mechanical loaders and scrapers, which revolutionized the mining of iron ore in the Birmingham district, where they were in use before they were installed in other iron-mining districts.

Particular attention has been paid to the installation of safety devices throughout all phases of the mine and surface plant.

The flowsheet of the crushing, screening, and washing plant indicates an efficient design, which subsequent operation has proved.

* Director, School of Mines, University of Alabama.

Limestone Mining at Ste. Genevieve, Missouri

BY RALPH W. SMITH*

(Washington Meeting, October, 1937; New York Meeting, February, 1938)

DEVELOPMENT of the lime industry in Ste. Genevieve County began in a crude way in 1840. According to information furnished by the Missouri Bureau of Geology, in the early days small vertical kilns built of stone were used, and wood was the fuel. Modern development began in 1904 when one company introduced steel shells lined with firebrick. Within a few years two more companies erected similar plants, and in 1925 a fourth company began operations. Thus there are today four lime companies at Ste. Genevieve, all operating at high capacity and with a gradually increasing production.

GEOLOGY

The limestone of Ste. Genevieve County, which has the chemical composition and purity required for lime burning, is the upper 80 or 90 ft. of Spergen formation. The Missouri Bureau of Geology, in its geological map of Ste. Genevieve County, gives the Spergen formation a total thickness of 160 ft., but only the upper portion is pure enough for lime burning, with an average raw-stone analysis of 98.7 per cent CaCO_3 ; 0.35 SiO_2 ; 0.3 R_2O_3 ; 0.35 MgCO_3 ; 0.3 SO_3 . The Spergen, one member of the Meramac group, Iowa series, of the Mississippian system, underlies an area approximately 12 miles long north and south, by 5 miles wide east and west (Fig. 1) in Ste. Genevieve County. The entire eastern two-thirds of Ste. Genevieve County is sedimentary, the formations dipping gently toward the east on an average grade of about 4 per cent, or $2\frac{1}{2}^\circ$. Thus the outcrops of the various sedimentaries run approximately north and south. A line running west from Ste. Genevieve crosses the outcrops of the sedimentaries, ranging from the top of the Mississippian to the Ozarkian geological system in a space of 12 miles. Thus with the gentle dip of $2\frac{1}{2}^\circ$ the 160-ft. thickness of the Spergen formation stretches out in its outcrop so that it is approximately one mile wide. The early quarry and lime developments at Ste. Genevieve were in this easily obtained surface rock of the Spergen outcrop.

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* Director of Research, Peerless White Lime Co., Ste. Genevieve, Mo.

Even as late as 1920, all limestone was removed in open quarries, some of which had been worked back into the hills (all of Ste. Genevieve County is rolling hill country) until the operators began to encounter some of the overlying St. Louis formation, which could not be used for lime burning because of its chemical composition. Cost of removing this undesirable stone increased extraction costs, and some of the companies

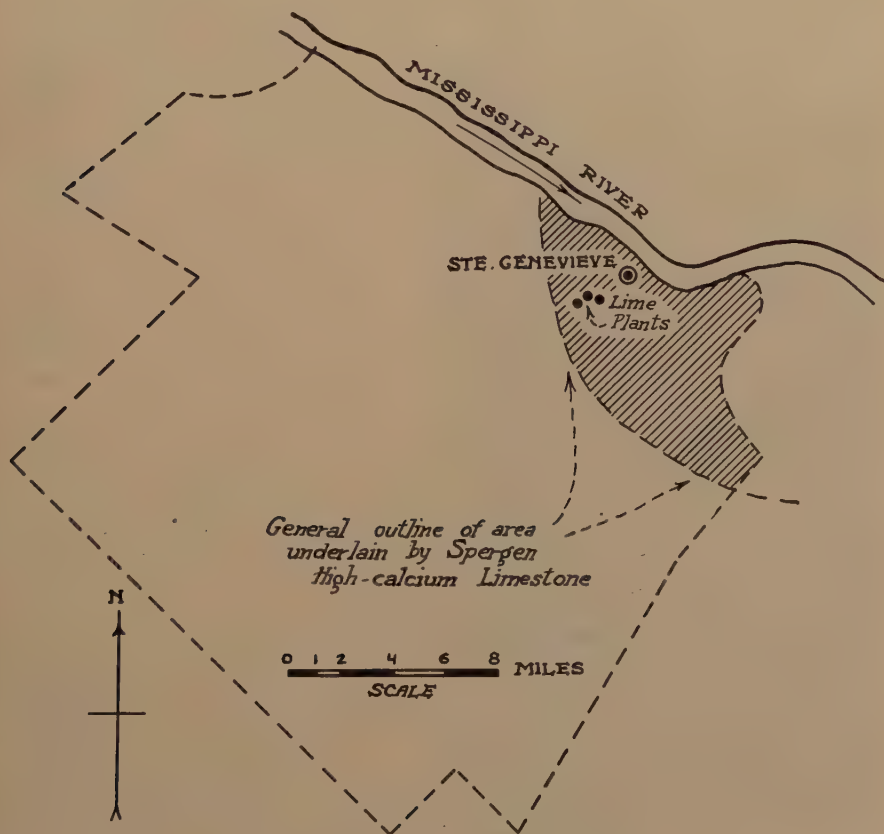


FIG. 1.—STE. GENEVIEVE COUNTY.

began to realize that there were advantages to underground operations. Mining started in 1921 in a small way. Within a few years better mining methods were developed, tonnages increased because of the excellence of the lime for chemical purposes, and today mining operations are extensive. Three of the four companies have underground workings, and one produces its entire tonnage from its mine. Average analysis of the burned lime from this district is 97.2 per cent CaO ; 1.7 CaCO_3 ; 0.3 SiO_2 ; 0.3 R_2O_3 ; 0.3 MgO ; 0.2 S.

MINING METHODS

Underground work was started by the Peerless White Lime Co. in 1921. At first long chambers with intervening long pillars, somewhat similar to ribs in coal mining, were used. With this method the percentage of useful stone left in the ribs was rather high, and as the demand for tonnage increased the number of available tracks for loading stone was insufficient. Therefore this type of mining was gradually changed over

to room-and-pillar method. In 1926 the general mining method was further altered to increase both the mining efficiency and the transportation methods. It took some years for the new combination of mining and transportation methods to swing into full effect, but the considerably higher efficiency now being obtained is full justification for major changes as planned during 1924 to 1927.

At present a much higher rock tonnage per man-hour of labor is obtained than in 1924, the tonnage of rock broken per pound of dynamite has about doubled (from 2.6 tons per pound to 5.0 tons over-all consumption), amount of other mine supplies (fuse, caps, etc.) and haulage power costs per ton have been correspondingly reduced. Much of this increased efficiency has been an absolutely necessary development since 1930 because of the extremely low market price of the finished products, but the efficiency was available because of



FIG. 2.—FINISHED PORTION OF PEERLESS MINE, WHERE IT MEETS OUTSIDE FINISHED OPEN QUARRY. ROOF IS 75 FT. HIGH HERE.

the long-time plan evolved during 1924 to 1927 and executed over a period beginning in 1924 and still in use.

At present the 80-ft. layer of high-calcium stone in the mine is being worked on two levels, the upper one about 30 ft. above the lower one. (A finished portion of the Peerless mine is shown in Fig. 3.) The original headings are driven on the upper level in the 15-ft. oolitic layer, which is slightly below the center of the entire 80-ft. thickness (Fig. 3). These headings are pushed considerably ahead of the other mine development

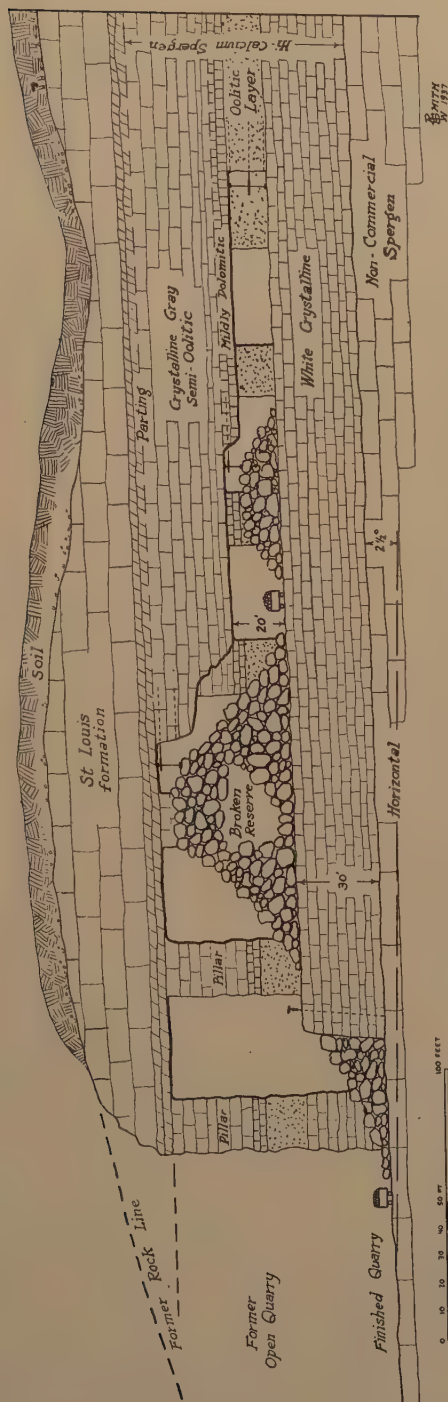


FIG. 3.—IDEALIZED CROSS-SECTION SHOWING MINING METHOD AT PEERLESS MINE.

(removal of 45 ft. of roof rock above the oolitic and subsequent bench mining of 30 ft. below). Since this company is using hand loading and operating only vertical stack kilns, which require stone 4 to 10 in. in diameter, it is desirable to keep the percentage of small "spalls" as low as possible, therefore the method of drilling and blasting is essentially different from that in most metal mines.

It is not difficult to keep the percentage of spalls satisfactorily low in open limestone quarries, where a combination of black powder, dynamite and large well-drill holes breaks huge tonnages, but that is somewhat difficult to duplicate in underground work where shooting in rooms, headings and drifts necessitates a heavier explosive charge per ton of rock and raises the spall percentage. In underground work the factors that affect the proportion of spalls are: (1) type and strength of explosive, (2) method of drilling, (3) method of loading and shooting the explosive, and (4) kind of stemming.

Explosives

In the Peerless mine, low-strength, bulky, ammonia dynamites give better results than low-strength gelatin dynamites. These ammonia dynamites had previously been tried in coal mines where it was found they increased the percentage of large lump coal. Gelatin is now used in the Peerless mine principally in wet holes or where there is heavy shooting in tight ground. Sometimes a few sticks of gelatin dynamite are inserted in the toe of a hole and ammonia dynamite is used in the remainder. The ammonia dynamite produces more lump rock than the gelatin because its rate of detonation is slower and therefore it approaches more nearly black powder. The approximate speed of these explosives was recently given by a technical representative of one of the powder companies as:

	FEET PER SECOND
Black powder.....	3,000 to 4,200
Ammonia dynamite.....	6,500
Twenty-five per cent gelatin.....	12,470

In using gelatin, considerable care must be exercised in properly loading the hole to produce a minimum of lump rock, because the margin of safe overload for good shooting is much smaller than in metal mines. Fig. 4 illustrates this condition. The effect of a slower dynamite is to extend the limits of loading within which it is possible to do good shooting. This also induces a condition that tends to waste some powder by overloading, but the expense of wasted powder is much less than the waste from increased spalls. Furthermore, this powder waste is probably more than offset by eliminating the necessity for reloading holes. In the effort to make good rock with a faster dynamite, it is not an uncommon practice to reduce the charge until the hole is underloaded, and this increases powder consumption rapidly. There are fewer holes that

do not break with proper loading of the slow dynamite. Fig. 4 also illustrates the large possibility of underloading a hole charged with black powder. A different spacing of drill holes or a lighter burden would change this, but this increased drilling cost and other characteristics of black powder (poisonous resulting gas) do not seem desirable underground.

The chart, Fig. 4, is merely an arbitrary and theoretical manner of illustrating why better results are obtained as far as spall percentage is

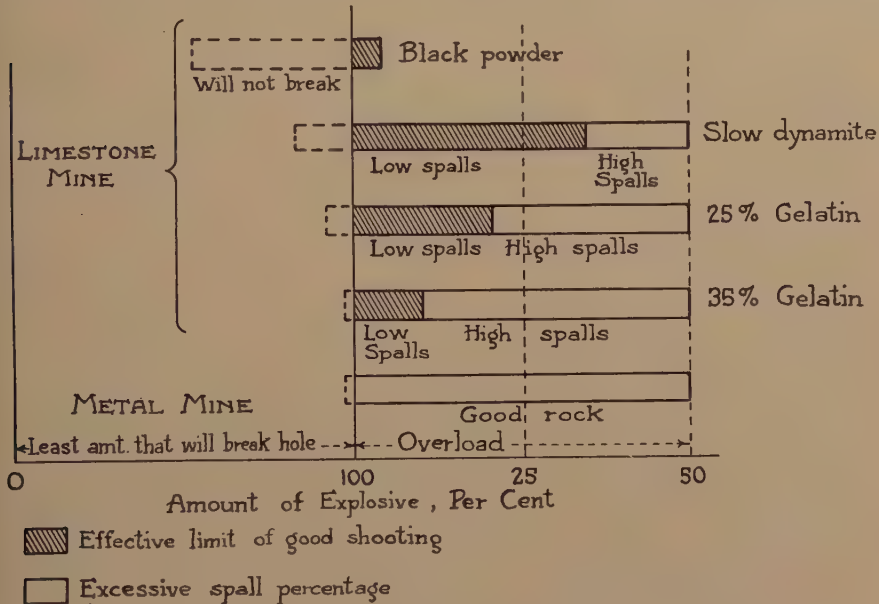


FIG. 4.—BLASTING OVERLOAD CHART.

concerned from "slow dynamite" than from other grades. It also gives a rough comparison with satisfactory blasting in the average metal mine. Since the chart is arbitrary and is based on observation instead of measured, scientific testing, it is labeled "approximate."

Starting with zero powder at zero per cent on the chart and gradually increasing the powder charge in any hole until it is just enough to break the burden, that amount of powder can properly be designated as a 100 per cent charge for that hole. This amount will vary with every hole according to its burden, depth, and distribution of burden in addition to location, whether top, middle or bottom. Each hole has a theoretical 100 per cent powder charge. With anything under this the hole will not break, or will only partly break, which is indicated by dotted lines on the chart. Thus, as we *recede* from a 100 per cent charge the powder becomes increasingly ineffective. The difference in the size of the dotted areas represents the relative danger of underloading with various types of

explosive. Thus, with black powder the area is large because it is difficult to get enough black powder in the hole to break the burden. This area is much smaller with the most satisfactory explosive, the "slow dynamite," but even then is larger than with faster explosives. Although this area is fairly small with the faster gelatins, this protection is expensive; as it is paid for by having a small area on the other side of the 100 per cent line.

Any dynamite in addition to the theoretical 100 per cent is unnecessary, but the practical powderman must be able to establish within practical limits the charge that will break the hole and yet not be an overcharge. His measurements of the hole may be exact and his judgment of the charge may be fairly accurate if there were no other variables; but there is one variable that cannot be exactly measured or estimated—namely, planes of weakness such as jointing or bedding planes. These often exert a considerable influence on the effect of the explosion and the proper amount of charge. If the hole is loaded with a fast dynamite an unseen plane of weakness may alter the estimated "correct" 100 per cent so much that it will be found after the blast that the hole was considerably overloaded. Waste of powder is the smallest factor in this case. It is the waste of limestone in spalls and the loading cost of moving them out of the way that are so important in the work considered here.

The crosshatching extends about to a 10 per cent overload for 35 per cent gelatin, to 20 per cent for 25 per cent gelatin, and perhaps to 40 per cent for slow dynamite. In other words, it is more difficult to overload a hole with the slow dynamite. It is also practically impossible to overload a hole with black powder; but, at the same time, it is almost impossible to get enough charge in the hole with standard drill spacing to bring it out. Thus the crosshatching represents the "effective limit of good shooting" because a powder within those limits for that particular hole will produce *low spall percentage*.

The white areas beyond the crosshatching indicate overcharge so great that a high spall percentage results. The portion of the chart from 0 to 100 per cent powder is on a different scale than that beyond 100 per cent. This was done to exaggerate the limits within which effective shooting could be done by the various types of powder. It is like putting a magnifying glass on that part of the scale, and this liberty was taken because the chart is only approximate and not based on actual measurements.

The portion marked "Metal Mine" shows that "good rock" can be produced anywhere from a 100 per cent charge to a 50 per cent overload. The explanation of this is that the material does not become waste because of its size after the blast. According to the chart, the powderman should try to attain the perfection that would enable him to load each charge at about 105 per cent. This would give maximum results with

regard to spall percentage with the least waste of powder. Powdermen at the Peerless White Lime Co. probably approach an average of between 105 to 110 per cent, over-all results.

A helpful device for improving the blasting is the shot record (Table 1), which was used to establish standard loads for various lengths and burdens of holes, and for checking results. If the management could always secure a powderman competent to make burden calculations on holes, a very high class of blasting with minimum powder would be assured. However, men of this ability are not usually available for this type of work. Ammonia dynamite partly solves this problem by making it more difficult for the average powderman to overload to the point of shattering the rock.

DRILLING AND BLASTING

Arrangement of Drill Holes.—Ammonia dynamite has the disadvantage of insufficient strength for some holes, especially in the toe. A common practice in the Peerless mine is to drill as many as nine rows of horizontal 18-ft. holes, one row behind the other, and if these holes were to leave a foot or so on each row as they are blasted in sequence, not over half the hole would be available for shooting by the time the sixth row was reached. This possibility is prevented by using one or two sticks of gelatin tamped well down into the toe of the heavier holes. Air spacing of slow dynamite is used on the lighter holes to prevent throwing of the rock; sometimes the sticks are pushed in until they touch but are not tamped to fill the hole; sometimes they are left with three or four inches of air space between sticks. A combination of all the foregoing practices for spall reduction has resulted in an over-all percentage ranging from 19 to 22 per cent, which is considered quite satisfactory.

TABLE 1.—Shot Record for Heading Holes

Location	Length, Ft.	Burden, Ft.		25 per Cent Gelatin		Other Powder Kind		Number of Bags, Tamp Size (1½ X 16)	How Fired	Results
		Toe	Collar	No. of Sticks	Length, In.	No. of Sticks	Length, Ft.			
No. 4 Top	15	4	2	2	12	8	5	6	Battery	Good
No. 1 Mid.	18	3½	3	2	12	7	4	5	Battery	Excellent—Left collar
No. 6 { Top Bot.	18	3½	2			6	4	5	Battery	Good
	18	3½	1			9	6	5	Battery	Too heavy } Fired together

Moist Stemming.—One of the most important means of securing uniform results and low consumption of powder is to use plenty of stemming and keep it uniform. The best results at the Peerless mine are obtained with a mixture of about two-thirds damp red clay and one-third pulverized limestone. Several years ago the spalls gradually increased during one of the summer months, and after much inquiry and search for the reason it was found that the clay for stemming had dried out during a drouth. The quality of blasting became normal as soon as the clay was moistened before mixing with the rock dust. This mixture is kept at a consistency that will form balls when squeezed in the hand. It is not uncommon after a blast to find holes in which a short portion of the collar is filled with this mixture and compressed by the blast until it seems as hard as the surrounding rock.

Blasting with Men in Mine.—Since it is more satisfactory from an operating basis (and also cheaper) to operate only one shift in the mine, all rock is mined on day shift and therefore it is necessary to do the blasting while the entire mine crew is at work. Electric caps are used entirely in the heading because this reduces lost time to a minimum where shooting is done during hours when the entire crew is at work.

Extreme safety precautions are necessary when blasting is done during regular working hours. The Peerless procedure is: The powderman and helper load the hole to be shot. The powderman then blows a signal on a loud air whistle centrally located near the headings. This warns the men of the coming blast and informs them in which heading the blast will be. As soon as the men are safely in the clear, protected by pillars, he gives two short toots with the whistle. This is the signal for the assistant to fire. The assistant, who has been waiting at the spot where the blast is to be, hears this signal and ignites a small amount of red-fire powder to burn near the hole for one-half minute and retires to a point of safety. Here he connects his wires to the battery and fires the shot. The entire operation requires about one minute if there is no unusual delay.

Red Flare as Safety Measure.—The red-flare powder (compounded by the Peerless company in its own laboratories) was adopted after a minor accident, which might have been serious. A few small rocks were thrown into a group of men who thought they were in the clear. These men were also seen by the head powderman, who thought they were not in the line of the blast. Had the red flare been set for the blast, the men would have seen that they were not safe and would have retired to another side of the pillar. The red flare has been effective a number of times since its adoption and has helped maintain speed of production with increased safety.

Original Headings.—Fig. 5 shows the method of placing holes and shooting the original headings driven in the 15-ft. oolitic layer. These headings are drilled with a light-mounted jackhammer drill, dry holes, from a 14-ft. column with three rows of horizontal holes, a top, middle

and bottom. Seven-eighth inch hollow hexagon collared steel with 3-ft. steel change is used. It is not unusual for one set of steel to last for a week of drilling before it is resharpened. This is because the high-calcium stone has a Mohs hardness of about 3.5, whereas the hardened steel bits are at least 6 by the same scale.

Firing Order and Loading Rock.—A special firing order and program of rock loading are used, to further reduce the percentage of spalls from what it would be normally in tight shooting necessary in original headings. Assuming that a face has been cleaned up and a set of holes is ready to be shot, the sequence of illustrations in Fig. 5 shows the order of firing the

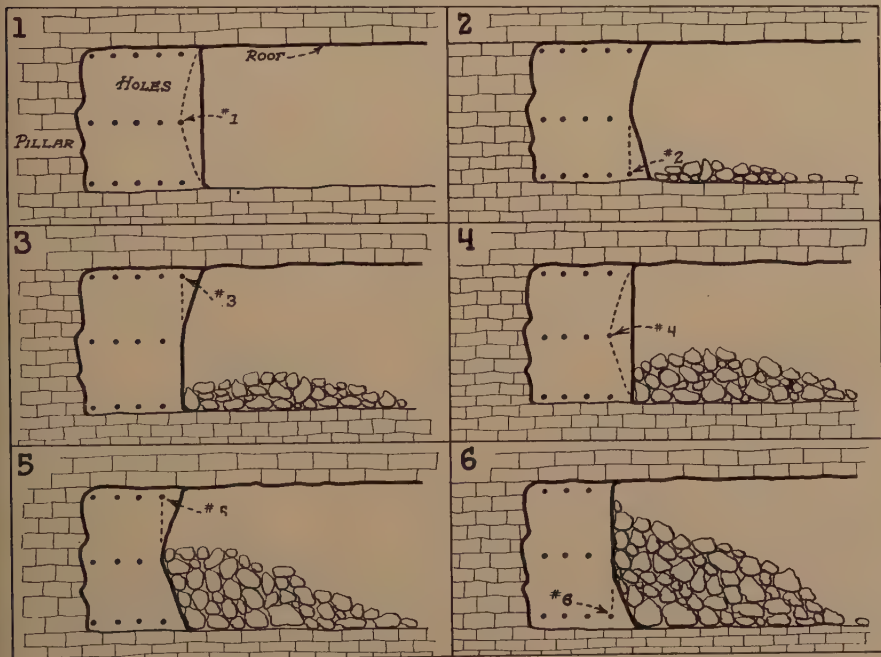


FIG. 5.—SHOT-FIRING ORDER IN HEADINGS.

shots in such a way that the broken rock does not add to the burden on the explosive. Hole No. 1 is fired first, throwing the broken rock on the floor in a low pile in front of hole No. 2. No. 2 is then fired, adding to the pile of broken rock in front of No. 3. After No. 5 is fired, the broken rock is piled up so high and so tight against the remaining face that this rock must be removed by the loaders before No. 6 hole is fired, getting ready for new drilling, or a new sequence of firing that will duplicate the foregoing. By this program no holes are fired against a rock pile when the heading is in the condition shown at hole No. 6.

At one time V-cuts were drilled and shot, especially in headings that presented a long face in which the rock line was straight or nearly so. This V-cut started a "break," against which later drilling could break

without undue rock burden, and consequent necessary heavy shooting with high resultant spall percentage. The technique of drilling and shooting these V-cuts was rather difficult, miscalculations on powder charges were easy to make and rather frequent in number, and so this practice was entirely abandoned. In present practice the driller sets up his column at a point where he can take advantage of any irregularity (usually present to a greater or less degree in every heading) remaining from the previous heading cut, and he drills back from this irregularity,

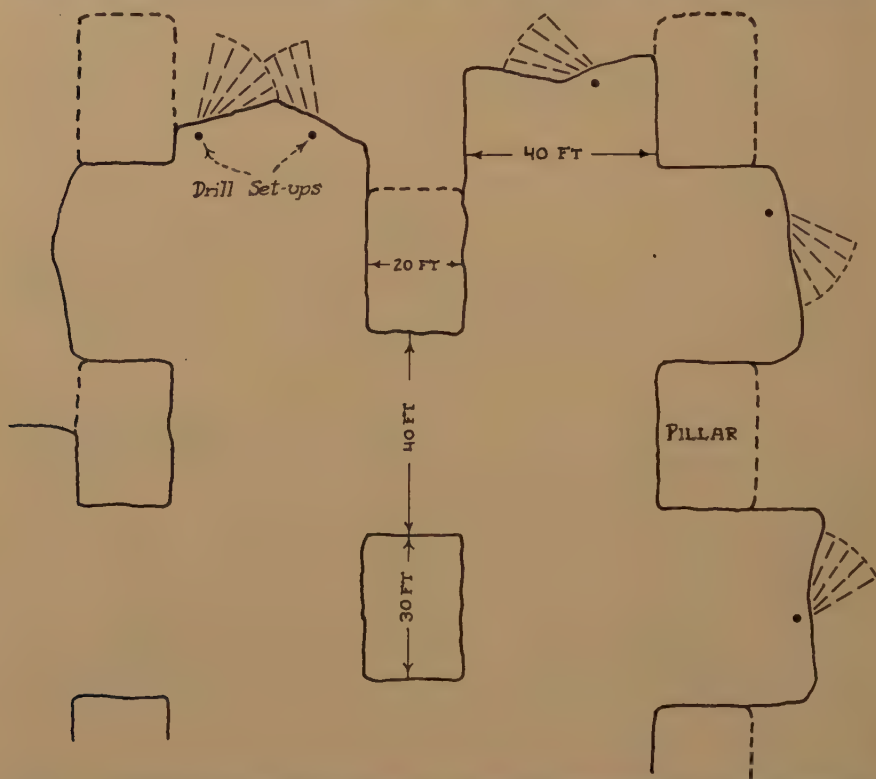


FIG. 6.—METHOD OF SETTING UP DRILL COLUMN IN HEADINGS FOR DRILL SWINGS.

swinging his holes farther and farther back from the face with each row, as indicated in Fig. 6. Thus from three to six sets of holes (horizontal rows of three: top, middle and bottom) are drilled from one setup (as per Figs. 5 and 6), the holes ranging from 12 to 20 ft. deep.

Complete removal by the loaders of the 15-ft. layer of oolitic layer as described above leaves ample room in which to perform the next mining operation; i.e., removal of an overlying layer of mildly dolomitic rock. Of this roof layer, 5 to 6 ft. in thickness is now shot down over a large area of the mine with flat holes. In starting this operation in an empty room 15 ft. high, a drill is mounted on a 12-ft. column (same as heading drill

and column) and two rows of holes, one above the other, 12 to 15 ft. long are drilled with a slight upward tilt into the roof as shown in Fig. 7. These extend across the heading as shown, three near the center, with two "trim-up" holes along the pillar lines. The ends of the top row of holes is calculated so that it will just penetrate through the dolomitic layer. When this set of holes is ready to be shot, the central three are shot simultaneously with electric battery; following this, the trim-up holes are loaded and shot electrically according to the burden left on

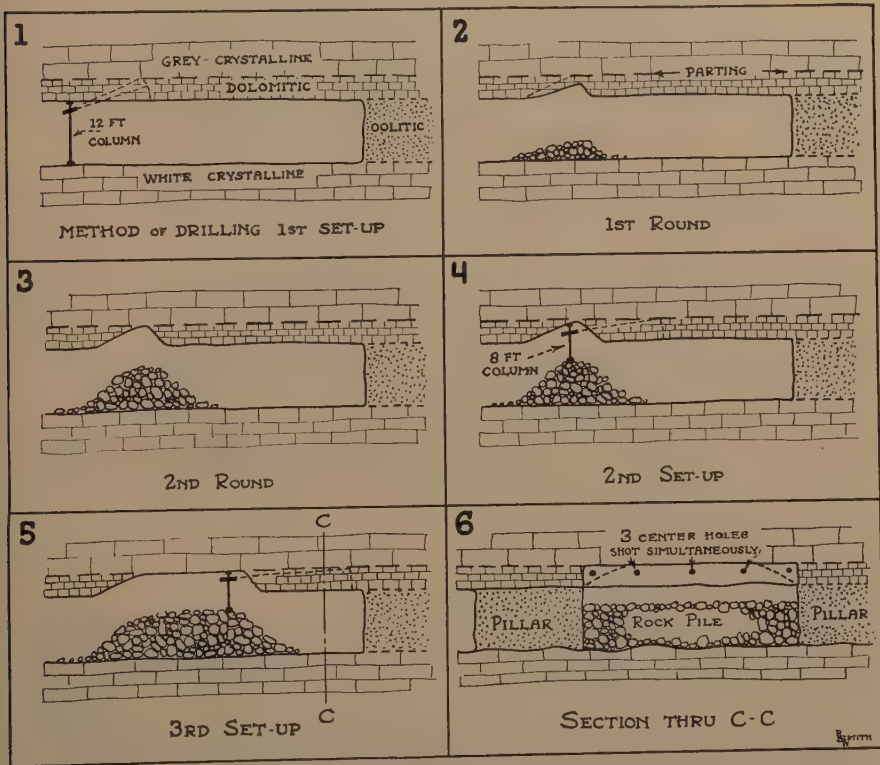


FIG. 7.—SEQUENCE FOR STARTING FLAT-HOLING DOLOMITIC LAYER.

them from the center-hole shooting. This initial "break" entirely penetrates the dolomitic layer at the toe of the holes and the broken rock furnishes a pile on which to mount a shorter drill column, and puts the drillers fairly close to their work for true flat holes. The 5 to 6-ft. layer of dolomitic rock expands when broken to fill the room (now 18 to 20 ft. high) with 10 or 12 ft. of broken rock so that column drills can be mounted on this rock pile.

These flat holes are drilled about 20 ft. deep, five holes to each 40-ft. room, three center holes about 12 ft. apart and semiparallel, with trim-up holes along the sides of the pillars. The three center holes are again shot



FIG. 8.—SIXTY-FOOT LADDER SUSPENDED BY ROPE GUY LINES FOR ROOF INSPECTION.
Top end of ladder is not touching roof or pillar.



FIG. 9.—CONSTRUCTION OF MINING TRAPEZE.

simultaneously with electric caps, and the trim-up holes are shot later with a dynamite load figured according to the quantity of rock left after the center shot. This method of dropping the roof up to the 20-ft. mark leaves a good safe roof because it breaks clean to one of several partings present. Thus with very little trimming the mine roof is in a good condition, under which men can work safely for the next mining operation. This layer of rock is now entirely removed, leaving an empty room 18 to 20 ft. high by 40 ft. wide, which is ample to allow for the swell of the remaining 20 to 25 ft. thickness of roof yet to be broken down. (Ladder for inspection of roof is shown in Fig. 8.)

High Roof Rock.—The remaining 20 to 25 ft. of roof rock is taken down by two methods, according to which one best fits the occasion. One method is with flat holes similar to the plan just described (see Fig. 3), and the other is with an 18-ft. column drill mounted under the 20-ft. roof. From this column drill a series of rows of vertical holes 20 to 24 ft. long are drilled in the roof. These are drilled in rows of five each, with about an 8-ft. front, and are fired electrically in the same way as the flat holes; i.e., the middle three first and trim-up holes later. Dropping this upper layer of roof rock fills the room to within 5 to 8 ft. of the roof, and this enables the miners to trim the roof and leave it in first class condition for later mining operations. Care must be taken to trim this roof carefully

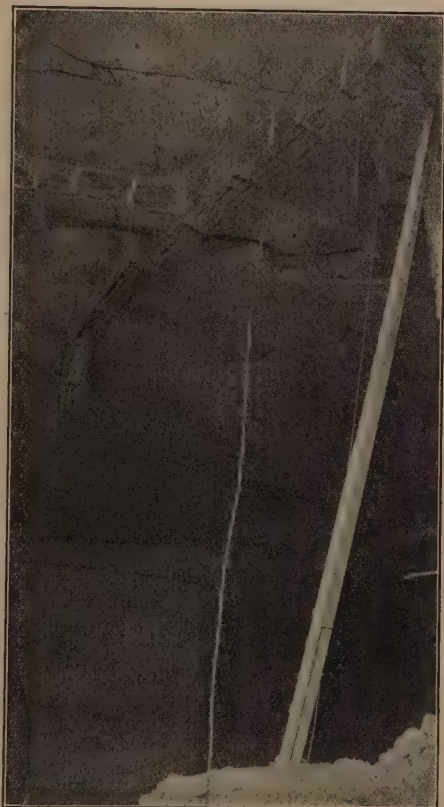


FIG. 10.—MINING TRAPEZE IN PLACE ON ROOF.

before the rock is removed, for it would be very expensive and in some cases impossible to do anything to the roof to make it safe after the room is empty. Portions of the mine are not overlain by a solid layer of the St. Louis formation. Consequently mud cracks and seams extend down through the roof into the Spergen formation, and are the cause of mining difficulties in places. Where these seams are not extensive, they are given a special treatment in the roof while the high rock pile is still in place. Holes are drilled on each side of the seam at intervals of 4 to 8 ft.,

into which expansion hooks are driven. Short pieces of rail are then laid in the hooks, and oak planks are laid on the rails so that the planks are close to the roof and directly underneath the mud crack. In this manner even a crooked mud crack is followed across the roof. If a small quantity of mud (containing gravel and chert nodules) falls on the planks, it quickly closes up the crack and the crack is thereby left safe for future mining operations after the stope is empty. Some of these roof repairs have been in place for 10 years and still appear to be in good condition.

Mining Trapeze.—An interesting development in the Peerless mine is the "mining trapeze" shown in Figs. 9 and 10. An area of roof about



FIG. 11.—THIRTY FEET MINED IN ONE BENCH.

80 by 100 ft. became unsafe because a supporting pillar had been removed. As it was very desirable to bench out the rock under this area in order to gain access to workings on the other side, it was necessary to make this roof safe for miners, loaders and locomotive men underneath. This was accomplished by the suspended catwalk as shown. This platform is built in sections one at a time as the miners advance underneath the roof. Expansion hooks 1 in. in diameter and about 2 ft. long are driven into short inclined holes drilled in the roof. Suspension rods (1-in. dia.) are then hung from these hooks at a reverse angle, so that when the suspension rods are fitted with a section of wooden platform it can be

pushed forward and fastened with spikes to the end of the platform already built. Thus one unit after another is added. These catwalks are rigid enough so that light jackhammers can be mounted on the platform, either on a drill column or a slide board. Thus if a piece of bad roof extending beyond the end of the platform is to be taken down, the drill is mounted at the end of the platform, the bad roof is drilled beyond the platform, and after the drill is taken down, the hole can be shot without any damage to the platform (the rock falls directly down, thereby missing the platform). This leaves good solid roof beyond the end of the platform, into which new expansion hooks can be placed so as to add another section of platform. Sometimes auxiliary platforms

are added branching away from the main platform. The entire trapeze runway is left in place so that the foreman can inspect the roof from time to time.

REMOVAL OF LOWER BENCH

After the high rock pile is removed, the lower 30 ft. of bench rock is mined (see Fig. 3). This lower bench mining is done with rows of vertical jackhammer holes, usually five holes for a 40-ft. stope fired electrically. During the early development of this lower bench system, the rock was removed in two benches of 15 ft. each with loading tracks on both benches. Thus there were three loading levels in the mine. After the benches receded far enough to build a 4 per cent haulage grade for locomotive haulage to the lowest level, the intermediate 15-ft. bench was stopped, and at present the entire lower 30 ft. is mined in one bench (Fig. 11) with one haulage-track system for the lower bench and a second for the original headings and upper rock. In mining this 30-ft. bench, a row of vertical jackhammer holes about 20 ft. deep is first drilled from the top of the bench and shot. This breaks off and leaves a rough lower ledge 8 to 10 ft. high and wide enough for the drillers, so they can follow up with a row of shorter bench holes to the bottom of the high-calcium stone. This lower shot is not made until the rock from the upper shot is entirely removed so as to reduce spalls and increase the tonnage broken per pound of explosive. Thus when this mining operation is completed, the total extraction is 85.6 per cent of the available rock, according to calculations based on one area unit, as shown in Fig. 12.

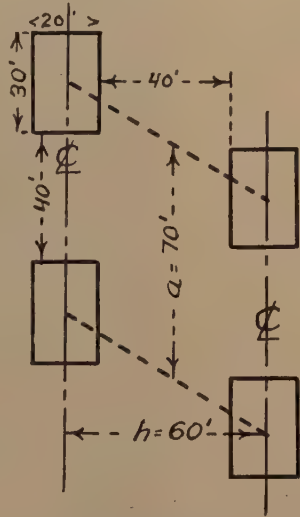


FIG. 12.—CALCULATION OF EXTRACTION.

Total area of unit:

$$\text{Parallelogram} = ah = 4200 \text{ sq. ft.}$$

$$\text{Total area of pillar portion} \\ = \text{one pillar} = 20 \times 30 = 600 \text{ sq. ft.}$$

$$\frac{4200 - 600}{4200} = 85.6 \text{ per cent}$$

LOADING AND TRANSPORTATION

All loading in this mine is hand labor on contract tonnage basis. Koppel standard "2 by 36 low rocker cars" (Koppel classification) with 36-in. gauge are used throughout. These hold about 3 tons of spalls and 5 tons of man-size stone. Two Whitcomb 8-ton gasoline locomotives are the motive power. These pick up train loads of six to twelve cars per trip. Outside of the mine, spall cars are switched out of the train and sidetracked until there is enough for a train to be run out on the spall

dump. All rock and spalls are weighed on a Howe scale and the weights punched on individual scale tickets for each car, one copy for the company and a carbon copy for the contractor. The rock is hauled to five different concrete rock bins at the kilns, where it is dumped by the haulage crew (Fig. 13). The mine has about 4 miles of track, including underground and outside track. Maximum haulage grade in the mine



FIG. 13.—CONCRETE ROCK BINS AT KILN.

is about 5 per cent, but most of the grades do not exceed 2 or 3 per cent. Thirty-five pound rail is used throughout, and most frogs are No. 4 plate riveted, 42 in. long with 12-ft. lead rails and 6-ft. switch points. An extra Whitcomb locomotive is on hand in case of breakdowns. Rock-storage bins at the kilns hold approximately 1500 tons when filled. This affords enough storage so that the kilns can be filled night and day over the 24-hr. period, whereas the mine operates only 8 hr. daily. Evolution of the transportation system is shown in Fig. 14.

USABLE SPALLS

A fair tonnage of spalls is furnished to the pulverizing department. The locomotives haul this over a steel trestle about 25 ft. high, where it is

dumped into bins that feed direct to a hammer-mill crusher from which it is fed by conveyor belt to an elevator and storage bin. This storage

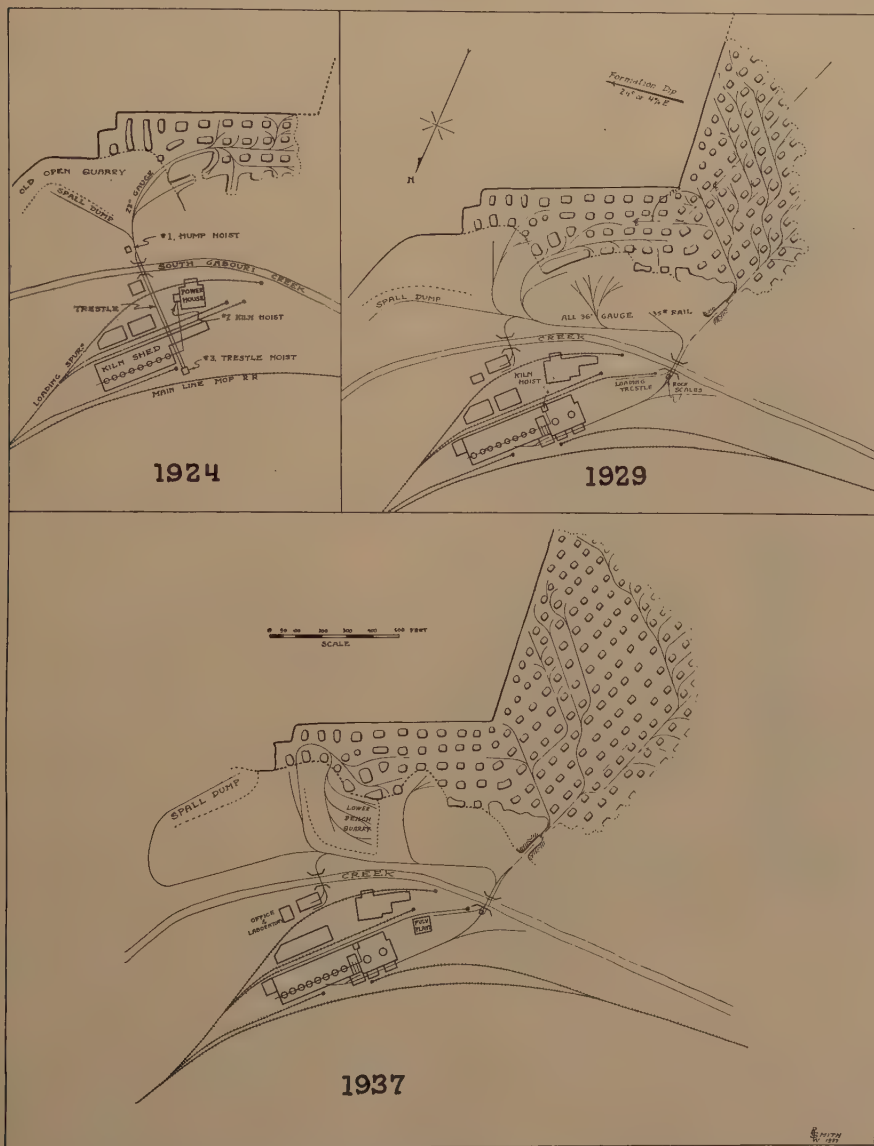


FIG. 14.—EVOLUTION OF PEERLESS MINE AND TRANSPORTATION SYSTEM.

bin feeds to a Raymond roll mill with coke-furnace drier. Products from this mill are sold for putty powder, calcimine, mineral feed fillers, pigment and rock dusting. This material averages 99.6 per cent through a 325-mesh screen.

KILNS

Two sizes of kilns are in operation at the Peerless plant (Fig. 15), six shaft kilns 14 ft. in diameter by about 60 ft. high and two shaft kilns 16 ft. in diameter by 90 ft. high. These kilns all operate on producer gas furnished by a remodeled heavy-duty Woods gas producer (Fig. 16). All kilns have induced draft furnished by exhaust fans. In order to operate in this manner, the kilns are built with automatic closed tops.



FIG. 15.—KILNS AT PEERLESS PLANT.

The two large kilns have two decks of combustion chambers, whereas the small kilns are fired on one level only. Kiln temperatures average about 2400° F., and the kilns are drawn and refilled every 4 hr. The lime is removed into draw carts and dumped on a shed floor, where it is cooled and hand-sorted for the various grades desired by the trade. A considerable tonnage is hydrated in a Kritzer continuous hydrator, after which it is treated in two types of Raymond air separators for removing bits of core, slag, or other plus 200-mesh grit. Average CaO content of the quicklime is 96 to 97 per cent, while the hydrate analyzes about the same figures in $\text{Ca}(\text{OH})_2$. This type of high-calcium lime has a very wide usage as a chemical reagent and is used in the manufacture of such products as baking powder, lubricating grease, paper (for causticizing and bleach liquor), casein glue, soap,

purifying gas, cold-water paints, sandlime brick, dyes, insecticides, calcium carbide, satin white (paper coating), tanning, steel, ore dressing, sugar, water softening, and many industrial chemicals.

LABORATORY

Since practically all of the output of this company is used as a chemical reagent, a chemical research and control laboratory was installed at the plant about 10 years ago, in which all shipments are tested for various chemical and physical specifications before they are allowed to leave the plant. Many interesting and valuable things relating to lime chemistry

have been developed in this laboratory, some of value to its customers, others of value to the company as control tests.

Another activity in which the laboratory played an essential part was the development of a plant for the manufacture of precipitated calcium carbonate, U.S.P. The Peerless company has been manufacturing and marketing this product since about 1930 and has developed it to such a high degree of control and purity that it has been necessary to enlarge the plant many times. Precipitated calcium carbonate is used principally as a base for tooth powder, tooth paste, face powder, and also in high-grade drugs and pharmaceuticals (as an antacid), in pigments, paper coating, paper filler, pill coating, and its use is being expanded to new fields from time to time. Peerless precipitated calcium carbonate has an average analysis of better than 99.0 per cent calcium carbonate, the remainder consisting of inert compounds of iron, aluminum, magnesia and calcium sulphate. As a result of the increased demand on the laboratory the company recently quadrupled its space and is now equipped to conduct a wide range of tests and experimental work relating to its field, but overlapping many other chemical processes. Many of these tests are extremely delicate (measured as low as one-half part per million). Some of the tests have been devised in this laboratory.



FIG. 16.—WOODS GAS PRODUCER.

ANOTHER LARGE COMPANY IN THE DISTRICT

The largest of the four companies operating in Ste. Genevieve County has mechanized its property extensively. In general, its stone deposit is somewhat more favorably located for mining operations, since the main body of stone is about $\frac{3}{4}$ mile east of the other deposits, and therefore about 100 ft. deeper on the dip of the stone. Thus the stone is deep enough so that no mud cracks are encountered in the mining operations. This deposit is tapped by a two-compartment incline shaft driven at an angle of 70°. The shaft is slightly less than 200 ft. deep, and the lower

end penetrates below the high-calcium stone about 40 ft., at the bottom of which is the drainage sump for the entire mine.

Hoisting Operations.—Broken stone is hauled to the shaft by gasoline-powered rubber-tired Mack trucks hauling 15 tons per load. Main haulageways from the rock piles in the mine workings are kept in very good condition, so as to speed up the rate of haulage and reduce wear and repair charges on the trucks. The main haulageways are worked over weekly with a standard caterpillar tractor with bulldozer blade similar to the type used on highway work. Rock is dumped from the Mack trucks by means of hydraulic piston dump bodies into a rock hopper large enough for one truck load, and the rock is fed from the hopper to a 60-in. Allis-Chalmers jaw crusher driven by Texrope with a 200-hp. electric motor. From the crusher the stone drops into a 25-ton pocket from which it can be drawn from either one of two air-operated gates into two 8-ton capacity skips in the shaft compartment. These two skips operate in balance at a hoisting speed of 400 ft. per minute, powered by Allis-Chalmers electric heavy-duty hoisting motors in the hoist house at the surface.

Mining Methods.—Room-and-pillar mining system is used exclusively in this mine. The pillars are spaced so that the rooms are 45 ft. wide and 45 ft. long between pillars on the pillar line. Rectangular pillars 20 by 30 ft. are standard, cut in a straight line in one direction and staggered at right angles to this line, so that a pillar always comes opposite an opening on the next pillar line. This arrangement reduces the sharpness of the bends for the main haulageways, so that the haulage trucks and mechanical loading equipment can be handled more easily.

The shrinkage method of mining is used. The first step in the operation is to mine out approximately 22 ft. in thickness of the bottom layer of the Spergen formation. This stone is entirely removed before the next operation begins, so as to make room for the broken rock swell. In some headings it is desirable to mine the overlying oolitic layer separately, in order to keep it segregated during the burning operation in the kilns. In these headings, following the removal of the bottom 22-ft. layer, the 14 or 15 ft. of oolitic stone is shot down by flat-hole drilling, holes approximately 20 ft. long. These holes are drilled with jackhammers on slide boards, which lie directly on broken stone obtained from the 30 ft. of roof rock directly above the oolitic. This is accomplished by advancing one of the broken rock piles from the high roof until it meets the oolitic breast that is to be mined. Mining of this upper 30 ft. is then retarded sufficiently in these headings so that the loose rock pile keeps the drillers close enough to the oolitic face to facilitate drilling of this layer. Following the removal of the oolitic rock, the remaining 30 ft. of roof rock is taken down in inverted benches 10 to 12 ft. high, with flat-hole drilling on slide boards in the same manner that the oolitic is mined. When the

top of the Spergen formation is reached there is very little clearance of broken stone from the roof, probably never more than 5 ft. Because of the depth of the deposit, there are no mud seams and this affords an excellent roof with very little "roofmaking" necessary. All drilling is contracted on a linear footage basis. Such trimming of the pillars as is necessary is done as the rock is withdrawn from the stopes. This company keep approximately 500,000 tons of broken rock in reserve.

The primary drilling in the lower 22-ft. cut at the bottom of the deposit is done with heavy drills, using $1\frac{1}{4}$ -in. round lugged steel, mounted on an 18-ft. column. Holes are drilled in series of four holes to the row; a top hole near the roof, two middle holes halfway down on the 22-ft. layer and directly underneath the top hole, and a bottom hole near the floor, also in line with the other three. In starting off a round of such holes in a breast, advantage is taken of any natural irregularity in the breast so that the first sets of holes form a semi-V which is gradually opened up as the holes recede from the center of the breast. Ten rows of such holes are drilled for a round, and this round is loaded and shot as a complete round, using electric delay caps with 10 delays; 20 per cent gelatin and No. 6 electric delay blasting caps are used. A single round consists of 500 tons of rock. In shooting, the pair of middle holes is shot first, and the trim-up holes follow.

Loading.—Rock from the 22-ft. original cut is shot heavy enough so that no blockholing is necessary. All loading in this mine is done with 2-yd. electric shovels, each shovel having a total electrical horsepower of 250.

Tipple House.—A modern Allis-Chalmers tipple house is at the mouth of the shaft. Rock from the skips is dumped into a 30-ton bin from which feeders convey it to perforated shaking screens. Each screen is 4 by 8 ft., containing 4-in. square holes. The fines or spalls that pass through these shaking screens drop into bins built high enough above railroad tracks so that when necessary they can be run directly into cars. Some of this spall material is crushed to 4 mesh for agricultural stone, some is crushed to 10 mesh for the glass industry, and some is hauled to a waste dump, where it will soon be reclaimed for rotary kilns now being erected. The oversize rock from the shaking screens goes to a shuttle conveyor, which transfers rock to the sized-stone bins. These shuttle conveyors are so arranged that at no time is the rock allowed to drop over 3 or 4 ft. The conveyor is moved forward or back as the bin is filled, to avoid excessive dropping and spalling of the sized stone.

There are two bins for this sized kiln rock, one for the oolitic layer and one for mixed rock, the bins holding 1200 tons total. From these bins the rock is tapped by Allis-Chalmers feeders direct into a train of side-dump rock cars, which run directly underneath the bins. There are six feeders for this operation and by means of the feeders each car is filled with about $3\frac{1}{2}$ tons of stone. There are eight cars in a train, which is pulled with a 100-hp. Plymouth gasoline locomotive. One train is

sufficient to haul the sized stone from the tipple house to the kilns, located about three-quarters of a mile west. The train runs about 20 miles per hour and operates twenty-four hours per day.

Kilns.—The train of eight filled cars is spotted at the foot of the kiln incline, where the locomotive picks up an empty train and returns to the tipple house. While the train is making its return trip to the tipple house, the filled cars are hoisted two at a time on an incline with a 75-hp. electric hoist to the top of the kilns, where another Plymouth locomotive picks the cars up, hauls them to the proper kiln and dumps them. The filling operation is continuous throughout 24 hr. of the day. Kilns are filled after each draw, which is a 6-hr. interval. This train of eight filled cars is dumped and ready for the return trip at the bottom of the incline by the time the next train of rock arrives.

The company has 20 stack kilns in operation in this part of its plant. The kilns are 14 ft. in diameter by about 60 ft. high, and operate by direct firing on grate bars. They produce an average of 22 tons burned lime per day.

Lime is drawn into portable steel boxes holding an average of 3 tons each, which are transferred with lift trucks, operated by storage batteries. The batteries are charged every 12 hr. at the company plant. Two lift trucks are sufficient to handle the 75 portable lime boxes.

In its early days of operation the company obtained its stone from a small open quarry adjacent to the kilns. This was a hand-loading operation, and so was discarded as soon as the mechanical plant was completed. The old quarry, however, is not entirely depleted and is considered as a "stand-by safety" in case of enforced shutting down of the mechanical plant. It has not been necessary to use it for this purpose for some years.

Rotary-kiln Plant.—This company has erected a new two-kiln plant for handling minus 4-in. material from its tipple house. One kiln went into operation early in February and a second kiln started up during the summer of 1937. These rotary kilns are Allis-Chalmers and Link Belt, 8 ft. in diameter by 300 ft. long with an 8-ft. stack 125 ft. high. They will handle crushed stone from 2 to $\frac{1}{2}$ in. and capacity is given at 175 tons of burned lime per day. They operate on pulverized coal, and all movement of material is mechanical. Ultimate plans for this plant include four rotary kilns.

CONCLUSION

The total tonnage of high chemical lime, lime products and various raw stone moving out of Ste. Genevieve County has been growing at a steady rate for 30 years, and (with occasional setbacks caused by such severe depressions as that from 1930 through 1934) it appears that the curve for total tonnage is still going up. Thus Ste. Genevieve is becoming increasingly important as a major producing and distributing center for high-calcium lime products.

Mining Marble*

BY GEORGE W. BAIN,† MEMBER A. I. M. E.

(New York Meeting, February, 1935)

METHODS of mining building stone of any sort are planned to produce as few fractures as possible, and present a strong contrast to methods of mining metallic ores, which must be crushed eventually and have fractures produced in as great profusion as possible. Shocks transmitted to a stone invariably make it more porous and more permeable and reduce its effective life, therefore methods of mining building stone involve as



FIG. 1.—THE SUPREME COURT BUILDING, WASHINGTON, D. C.

This building illustrates the fine detail of workmanship, unprotected sharp corners and thin edges, and tall narrow columns characteristic of marble structures. These features require exceedingly coherent stone, and preservation of these features for decades or centuries demands many special quarrying precautions to keep absorption to 0.14 per cent or less, a detail not essential to stone for coarser architectural types.

few shocks as possible. This is singularly true for marble because marble structures are intended to have long life, greater delicacy of line, more gracefulness and fineness of sculpture than those made from any other structural material (Fig. 1). So that delicate lines may remain in almost perfect preservation for decades, or even centuries, it is important

* This article has been prepared with the cooperation of the entire quarry staff of the Vermont Marble Co., and is based upon the company's records of some of the largest underground quarrying operations undertaken on this continent. The author's chief duty has been to assemble the manuscript and call for specified data that are of importance to the quarrying industry in general. Manuscript received at the office of the Institute Jan. 1, 1935. Issued as T.P. 626, June, 1935.

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that the stone be insulated from violent shocks at all stages of handling. This precludes free use of explosives and too free use of wedges.

Marble is stone of pleasing appearance and capable of receiving a high polish, composed of minerals or mineral grains having essentially the same hardness. Many products marketed under the name of marble are not crystalline limestone; verde antique marble is chiefly serpentine and Belgian Black marble is an ordinary limestone which has no pore space. Many color effects and structural effects, adopted to give a pleasing appearance, require the marble to hold a sharp edge and resist crumbling, so that the stone must have a high degree of coherence. Any marble belt, whether it be Georgia, Alabama, Pennsylvania, Massa-

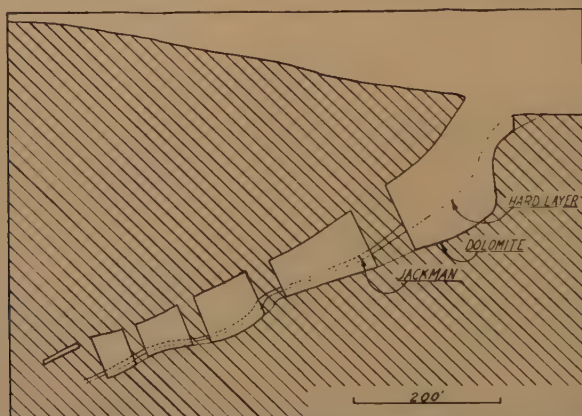


FIG. 2.—CROSS-SECTION OF MAIN QUARRY, WEST RUTLAND, VERMONT.

This shows the dip of the marble beds changing from almost vertical at the surface to a gentle eastward inclination above 180 ft. depth. The change occurs nearer the surface in the upper beds than in the lower.

chusetts or Vermont, is dotted with abandoned workings; rarely has the deposit been exhausted; usually the deposit has been abandoned because the marble is unsatisfactory. The few productive quarries lie along very limited zones or "veins" having stone with pleasing color, coherent grains and resistant to wear and weather. Formerly selection of zones with suitable stone was a result of expensive experience but more recently laboratory tests have proved slightly helpful.

Cost of underground operations exceeds open work by the total expense of driving a tunnel floor—an item ranging from \$2 to \$3 per square foot of quarry floor, value of marble which must be left as pillars, fixed lighting cost, cost of several hundred yards of subsurface standard-gage railroads, and numerous short-boom derricks. One reason for incurring this added expense in the Main quarry at West Rutland, Vermont, was that the only zone of stone worthy of the name marble turned from a vertical dip to a gentle eastward inclination at about 180 ft. depth (Fig. 2). The only course was to follow the marble. The

Covered Kitchen quarry at the south end of the West Rutland deposit is overlain by 50 to 80 ft. of water-bearing sand and clay and a surface zone of sheeted rock; it was necessary to tunnel southward in marble from the older workings and leave a roof to hold up the water-bearing overburden. At Danby, it was less expensive to drive a tunnel above the flat dry marble deposit than to quarry 100 to 200 ft. of waste overburden. Underground quarrying is adopted usually for one of the three reasons listed above. Procedure and expense are additional to that for ordinary marble quarrying.

QUARRYING PROCEDURE

No explosives are used in the actual quarrying of marble; all stone is cut loose by channeling machines or, less often, by other cutting devices. The plan of a quarry floor is laid out usually with the length along the strike of the deposit and the width across it. Longitudinal channels are cut first; if a valley or open seam occurs along one side of the deposit the channel most distant from this opening is cut first to release strains inherent in all rock. Channels are cut from side to side of the deposit at intervals equal to the width of the blocks desired; usually they are 7 ft. deep and 6 ft. apart. Reasons for occasional use of other spacing, and attendant expense, are given later.

The Key Block and Keyway.—The first block from a quarry floor is called the keyway key block; it is located with its surface nearly normal to the line of pull from a near-by derrick and preferably adjacent to one quarry wall (Fig. 3). If all key blocks can be located at a vertical quarry wall, a longitudinal channel is cut at about 5 to 6 ft. from the wall, or if the key blocks are in the middle of the quarry, two parallel channels are cut 5 to 6 ft. apart down the length of the quarry. A pull from the derrick is applied to the keyway key block and one side is wedged or pried sidewise until it breaks free, usually near the bottom of the channels. This strip, which is the keyway, has other channels at suitable intervals; holes are drilled under the blocks and they are raised one at a time with wedges driven into the holes. Holes are drilled under the strips on either side of the keyway and the key block for that strip is lifted first. The other blocks are not channeled free but are wedged off along lines of holes drilled with a gadder.

Modern gadders are equipped with pneumatic drills with pneumatic feed. Holes are drilled dry. One-inch hollow hexagonal steel with a $1\frac{3}{8}$ -in. gage rose bit is used. The drilled hole is about $1\frac{1}{2}$ in. in diameter. Wedges to fit these holes are made in three parts; the outside parts are tapered segments of a cylinder and the central part is a flat wedge (Fig. 4). The three parts are inserted in the hole as a unit. The thin edges of the segments project slightly from alternate holes; other wedges are tapped

to the bottom of the intervening holes and all wedges are driven in with a "bunter" bar.

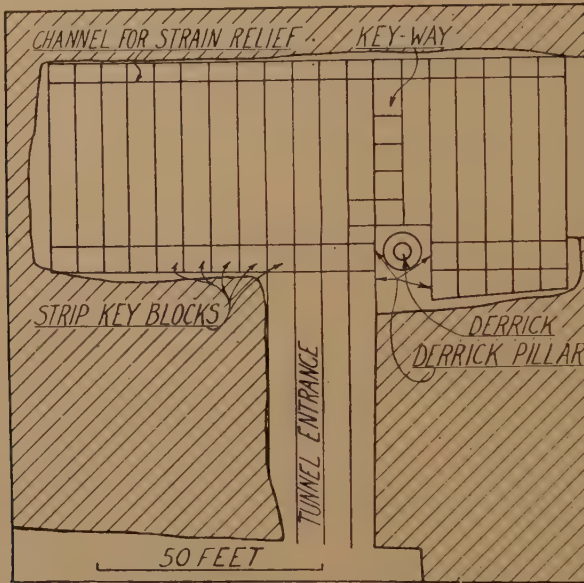


FIG. 3.—PLAN OF PART OF WEST BLUE QUARRY NO. 4, WEST RUTLAND, VT., SHOWING POSITION OF KEYWAY AND KEY BLOCKS.

The keyway lies so that the derrick pull and the channels are parallel. This prevents binding of the blocks against the channeled walls of the keyway. Strip key blocks are along a line at right angles to the keyway and are bounded by vertical channels.

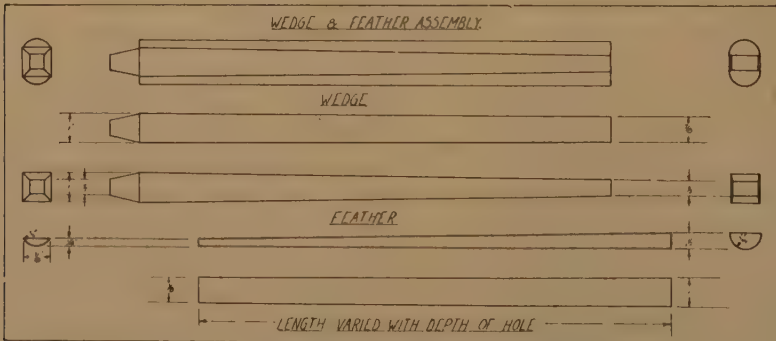


FIG. 4.—DESIGN FOR WEDGES AND FEATHERS.

End dimensions of both wedges and feathers are the same, irrespective of their length. The length of the assembled units varies from 1 to 4 ft. Wrought iron or mild steel is used for the feathers.

Working Force.—A 50-ft. quarry frontage, such as that between pillars in the Imperial tunnel at Danby, requires four channeling machines for efficient operation. Two channelers make the wall cuts and two

divide the floor into strips 5 to 7 ft. wide. Two gadding machines are needed for drilling off blocks and one jackhammer is required for general purposes. The quarry working force averages seven men per channeler.

Any quarry needs two channeling machines with a complement of four men, one gadding machine and operator, one drill-sharpening outfit and operator, one man to operate the power for the derrick, four laborers and a foreman. Such a force can produce blocks weighing up to 25 tons, and did operate the St. Alban's Bay quarry in northern Vermont throughout its life; although at that quarry drill and broach tools were used, not channelers. A force of this size operated the Brandon Italian quarry at Brandon and the Florentine Blue quarry at Pittsford for a number of years, but economical work requires more extensive operations.

Rate of Operations.—Channeling machines will cut 6 to 10 sq. ft. per hour of running time (Fig. 5). However, this is only part of the actual time; laying track and moving machines is an important item on the machine runners' report. A rate of channel cutting approximating the capacity of the machine requires a quarry floor large enough to keep the channelers out of the way of the gadding machines and derricks pulling blocks. Actually channelers average 4 to 8 sq. ft. per hour.

Factors Affecting Rate of Cutting.—Occasionally quarry floors are inclined at more than a 10 per cent slope and the machine is operated with a balance car. Laying the track for the balance car requires extra labor and the over-all rate of cutting is only 80 to 85 per cent normal speed. Where a channel must be cut in a horizontal position, as in tunneling, the rate is only 50 to 60 per cent normal speed.

Channel cuts more than 7 ft. deep are required for supplying oversize blocks and changing the slope of a quarry floor. The expense increases almost in a geometric ratio with the depth of the channel. The cut is started with a channel steel of wider gage; the cut deviates from a plane as it is deepened and wall friction and greater inertia of the cutting bit increase to such degree as to minimize measurably the force of a stroke. Taking into account increased steel breakage, power and labor charges, the cost of deepening a channel from 7 to 10 ft. and from 10 to 12 ft. is about the same as for cutting the first 7 ft. Channels as deep as 18 ft. have been cut, but they are expensive. Marble saved by deep channels in Pittsford Valley, West Rutland and Danby quarries compensates partly for the added quarry expense.

Other Methods of Cutting.—Wire saws using 1300 to 1500 ft. of wire were tried in the covered part of the Main quarry, West Rutland, during 1927-28. They were unsatisfactory because the stone is so coherent that the wire wears out before a cut is bottomed. Insertion of new wire is extremely difficult. Expense of operations was increased by the necessity for impounding used sand, an extremely difficult procedure in these inclined quarries, which must be kept developed to varied stages so as to



FIG. 5.—CHANNELERS CUTTING FIRST FLOOR IN DANBY TUNNEL.

One channeler is mounted on track on upper level at left and another on lower level at right. Length of channel steel, leaning against wall at left, and depth of channels may be estimated by comparison with the runner's helper. Bottom of holes of drill and broach tunnel channel appear on wall of pillar at middle right. Spacing of blasted holes in tunnels shows in the wall back of upper channeler. A dismantled gadder stands in lower foreground.

be ready to produce from any of the 22 beds in the deposit. Another design for a wire-saw set-up was tried during 1932-33 in the Covered Kitchen quarry, where the beds are vertical and any required type of marble can be quarried without removing those adjacent to it (Fig. 6). Nearly 3500 ft. of 0.25-in. wire was wound on the pulley system extending across the quarry. Satisfactory results were obtained. A wire speed

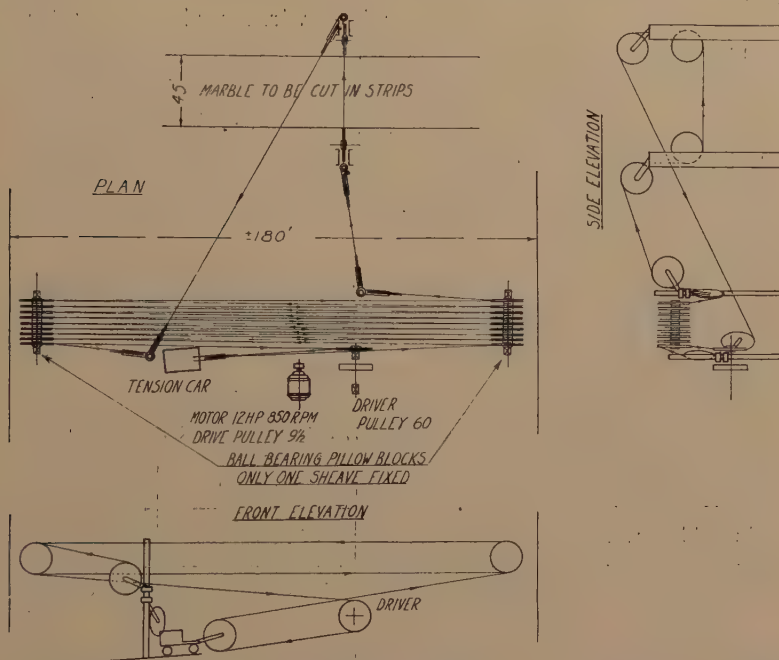


FIG. 6.—WIRE-SAW SET-UP IN COVERED KITCHEN QUARRY, WEST RUTLAND, VERMONT.

Approximately 3500 ft. of $\frac{1}{4}$ -in. wire is on the idler and standard sheaves of this wire saw. The idler sheaves are in the chamber (second from the south) between two rib-type pillars of the quarry. (See Fig. 13.) The marble strip to be channeled is 45 ft. wide and has keyway strips removed on either side for the saw standards. Sand used by the saw is impounded in these keyways.

of $17\frac{1}{4}$ ft. per second was used and wire was discarded when it was worn below 0.20 in., to avoid excessive breakage trouble at this lower gage. A summary of saw operations between Dec. 13, 1932 and June 15, 1933 is given in Table 1. Costs for a wire saw operating under very favorable conditions are contrasted with average expense for channelers in Table 2.

Channeler costs run 8¢ per square foot lower than for the wire saw. This difference is almost an average figure and varies both ways with quarry conditions. The setting-up cost for the wire saw is prohibitive for short channels or for a few; on the other hand, channeler costs increase in a geometric ratio with depth of cuts, by contrast with wire-saw work, which remains unchanged. The ratio of depth to length and number of

strips will be always the determining factors whenever wire saws come into more general use. Channelers cannot be dispensed with entirely; they are essential to cutting keyways for saw standards and sand catchment basins; greater mobility renders them indispensable for short cuts, especially around key blocks. Higher initial cost of channelers, while significant, is not as important as it might seem and is written off in about three years of use. Another apparently insignificant item is lack of induced strains along channels made with a wire saw; butts of "stunned" stone are lost adjacent to channel cuts.

TABLE 1.—*Time and Materials Record for Wire Sawing*

Time, Ft. per Hr.	Materials	
AMOUNT CUT, 2396.26 Sq. Ft.		
Running time, average.....	5.31	4.13 ft. of wire per sq. ft.
Working time, average.....	3.62	0.42 cu. ft. of sand per sq. ft.
Workman's time, average.....	3.29	
Including set-up, average.....	2.67	
Running time, maximum.....	7.06	

TABLE 2.—*Costs of Operating Wire Saw and Channeler*

	Wire Saw	Channeler
Original machinery.....	\$1373.89	\$7500.00 (appr.)
COST PER SQUARE FOOT		
Wire.....	\$0.079	
Channel steel.....		\$0.014
Sand.....	0.014	
Steel sharpening.....		0.016
Labor setting-up.....	0.084	0.027
Labor operating.....	0.227	0.198
Other operating.....	0.074	0.173
Total.....	0.478	0.418
HOURS LABOR		
Setting-up.....	0.220	0.058
Operating.....	0.393	0.342

The drill and broach method has proved advantageous for driving horizontal channels in tough hard rock of the tunnel cuts in the Danby quarries and has met with some success in verde antique development. A series of closely spaced parallel holes is drilled along the line of the proposed channel. Space between the holes is broken out with a broach-

ing tool. The channeling speed is about 5 to 7 sq. ft. per hour running time and 3 to 5 ft. per hour over-all.

The radiolax is used to cut vertical channels into a quarry wall (Fig. 7). This machine is a pulsating piston drill mounted on a swivel operated by a gear, so that it can be swung up and down in a vertical plane. The bit is moved up and down to cut a channel instead of a



FIG. 7.—RADIOLAX CUTTING KEY BLOCK IN WALL OF COVERED KITCHEN TUNNEL, WEST RUTLAND, VERMONT.

The radiolax has a pulsating piston direct connected to the cutting tool. Top of pulsators operating piston appear beneath staging for machines. Finished tunnel is illustrated by Fig. 11.

circular hole. One man operates the radiolax and can average 5 sq. ft. per hour running time, or 3 sq. ft. over-all in the West Rutland deposit. Work is much slower in the Pittsford Valley deposit. The machine is not used at Danby.

MINING OPERATIONS

The West Rutland marble deposit turns from vertical to a gently inclined attitude at a depth of about 180 ft.; even the vertical part of the deposit pitches down under 60 ft. of water-bearing soil at the south end. At Danby, 100 to 200 ft. of worthless rock overlies the commercial marble. Cost of removing overburden in each case exceeds the value of the marble available afterwards. The only economical method of approaching the marble is by a tunnel high enough for channeling oper-

ations. Marble pillars are left at intervals to support the tunnel roof. Tunnels above horizontal and gently dipping marble deposits are driven in waste and no attempt is made to save the stone. Tunnels above vertical deposits are driven in marble and the stone is saved wherever possible. The method of procedure is different for these two positions of the marble. The first kind of tunnel is driven at Danby and in the bottom of the Main quarry at West Rutland; the second at the Covered Kitchen quarry at West Rutland, in the Pittsford Valley quarries and in the Brandon Italian quarry.

Tunnel Driven in Waste

Early quarry work at West Rutland indicated that it would be necessary to make a tunnel floor on which to lay track for channeling machines. Quarrymen knew from experience that if the tunnel were driven by blasting out the waste rock, the marble below would be shattered and worthless. This marble must be insulated from the force of any blasts, and pillars must be left to support the rock cover and prevent rock falls due to widening of bedding planes and joints in the roof. The essential problems of mining marble were apparent but the quarrymen overestimated the strength of pillars and the manner of failure and underestimated the tendency of rock weaknesses to open.

Space to be tunneled out by blasting is insulated from the rock to be saved by a channel cut. The channeler is mounted on its track and the head with the cutting tools is inclined at a dip sufficiently steeper than that of the marble to meet the latter at a depth of 8 ft. (Fig. 8). Rate of channeling this 8-ft. cut is only about 65 per cent that for vertical cuts. Waste between the channel cut and the marble is raised with wedges and removed in waste boxes. Holes are drilled into the waste above the channel cut at intervals of slightly more than one foot. Each hole is loaded with a cartridge containing about one pound of black powder. The bottom holes are blasted first; if the cut is not fairly clean, it is mucked out before the higher holes are loaded. The tunnel is 7 ft. high and is kept about 50 ft. in front of production operations. Formerly waste was removed in wheelbarrows rather than with scrapers, or in waste boxes, and the tunnel was kept only 30 ft. in advance of quarry work.

This scheme is modified slightly for the drill and broach method used at Danby. A cut is drilled and broached 3 ft. deep; the first row of holes above the cut is blasted and cleaned out. A second 3-ft. channel is drilled and broached back of this cut and the roof of this channel also is blasted down. The remainder of the roof holes are blasted and the floor wedged up. The advantages of the two stages of channeling are: (1) it is easier to withdraw the broaching tool from a 3-ft. hole and very

difficult to pull it out of a 6-ft. hole, and (2) 3-ft. holes can be drilled without changing steel.

Rock is left at intervals to form the tops of pillars. Channels are cut around this rock remnant as each floor is quarried out below the tunnel floor (Fig. 9). No attempt is made to recover marble in the pillars; these serve to support the roof and soil cover, and to hold jointed roof rock in place. Some few joints are inherent to the deposit but a number are developed by tunneling and by rock expansion effects after the tunnel has been driven. Occasionally an unexpected, dipping, joint plane

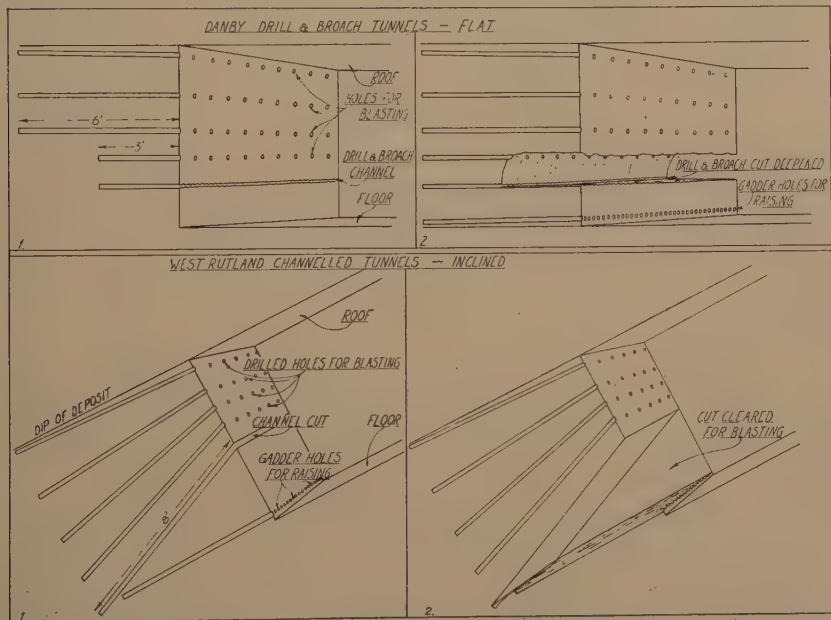


FIG. 8.—DRIVING TUNNEL IN WORTHLESS ROCK. ABOVE, AT DANBY; BELOW, AT WEST RUTLAND.

Note that cut is cleared at a single operation when channelers are used but is done in two stages when drill and broach cuts are made.

requires an extra pillar not allowed for in the original plan. Deposits free from joints, as at Danby, have pillars spaced rather far apart, but if joints are frequent, as near the Well Hole opening at West Rutland, pillars must be more closely spaced. Soundness of the quarry roof exerts greater control over pillar spacing than does thickness of rock cover.

Tunnel Driven in Marble

Nearly all marble deposits have a highly sheeted surface zone extending down 40 to 80 ft. Even the most careful core drilling and geological work cannot foretell with certainty whether marble below this surface sheeted zone has good quality and soundness. Accordingly, a small

quarry, 25 by 40 ft., is carried down from the surface to below the sheeted zone. Cost of a small prospect of this sort varies from \$60,000 to \$100,000; if satisfactory marble is found a tunnel is extended laterally in the marble and the real expense begins.

The tunnel is driven by mounting the channeler head horizontal and making the roof cut first (Fig. 10); this extends all around the prospect



FIG. 9.—PILLARS IN IMPERIAL TUNNEL AT DANBY, VERMONT.

The photograph gives some impression of the size, shape and multiple use of these pillars. Items of quarry equipment shown are: standard-gage railroad (left), derrick power (to right of central pillar), tramway for steel (above derrick power), and pulling and loading derrick (right).

quarry. The prospect is deepened one floor and another horizontal channel made parallel to the first. These horizontal channels are joined by a vertical radiolax cut at the ends. Another radiolax cut is made about 6 ft. from the corner of the quarry, thus freeing a key block on all except the back face. This key block is removed as though it were in the quarry floor instead of in the wall. The other blocks in the tunnel are drilled off with a gadding machine and wedged free (Fig. 11).

Favor for tunneling is divided between the channeler and radiolax combination and the drill and broach method. Generally the drill



FIG. 10.—CHANNELER CUTTING WITH HEAD HORIZONTAL.

Head is inclined slightly upwards to recover height lost by starting cut down necessary working distance.

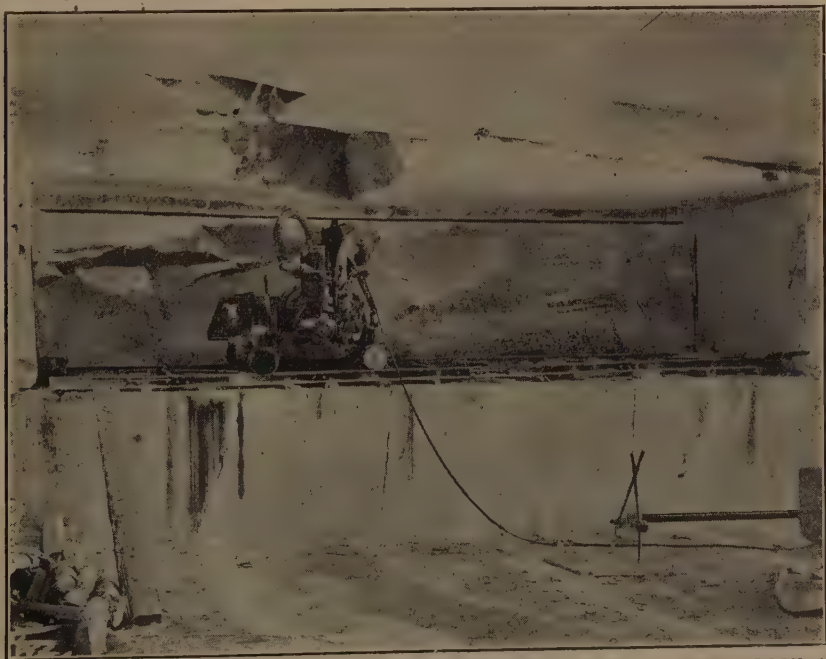


FIG. 11.—TUNNEL IN MARBLE, SHOWING POSITION OF RADIOLAXED KEY BLOCK AND OTHER GADDED BLOCKS.

Position of radiolaxed key block shows at extreme right. Gadder holes show back of next block to left. Blocks farther to left broke out on a series of joints or heads.

and broach method has a slight advantage in hard, tough rock and the channeler has a slight edge in softer, less coherent types.

Tunneling Costs.—Quartz bands in the roof of the Danby deposit cause the greatest fluctuation in tunnel cost. These impure zones do not vary greatly at West Rutland and cost shows a narrower range. Unsoundness necessitating change in the pillar-spacing plan, shorter runs for the channelers, more care in blasting close to pillars, and minor incidental items raise the cost above the minimum figure. Tunnel expense at West Rutland ranged from \$1.50 to \$2 per sq. ft. of tunnel floor during the past decade. The same cost varied from \$2 to \$3 at Danby. It was \$2.66 during 1932, and was distributed as in Table 3. No powder is used when the tunneling plan calls for saving the marble, but the labor and power charges increase for the extra channels and slower rate of cutting over drilling.

TABLE 3.—*Distribution of Tunnel Cost*

	PER CENT
Labor (general).....	34.4
Drilling.....	19.16
Drill and broach.....	15.8
Labor (total).....	69.36
Overhead ^a	25.34
Powder.....	3.95
Repairs.....	1.25
Total.....	100.00

^a Overhead includes employees' pensions, insurance, taxes, accident benefit, superintendence, rent and depreciation.

Pillar Spacing

The roof area of the Danby quarry is 13.2 acres and of the West Rutland workings 17 acres. Furthermore these underground quarries extend as far laterally as longitudinally. They have derricks and standard-gage railroads from the entrance to the farthest corners; the roof above them must be supported and remain supported all during the life of the quarry. Pillar spacing has been a serious problem from the inception of underground operations, although it does not seem to have been considered so by some of the early operators at West Rutland.

Spacing in Inclined and Horizontal Deposits.—The West Rutland deposit was worked originally by many small operators and was consolidated under one management about 20 years ago. Each operator quarried up to within 10 ft. of his property line—and sometimes even beyond his own line—so that the pillar spacing corresponded to the property lines. These boundary pillars extended as a curtain from hanging wall to footwall, were very thin and often more than 100 ft. apart.

Extension of underground operations left the roof more than ever dependent upon pillars for support; these boundary pillars have crushed

badly and scaled along shear planes running into them from either side. This warning resulted in more numerous square pillars, systematically spaced in the gently inclined covered part of the Main quarry (Fig. 12).

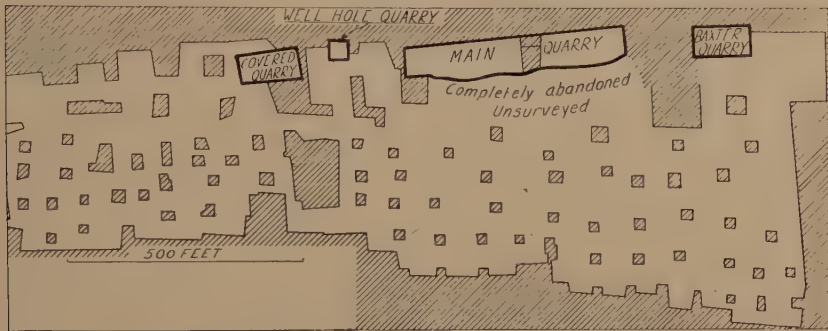


FIG. 12.—PLAN OF PART OF COVERED KITCHEN QUARRY, WEST RUTLAND, VT., SHOWING EVOLUTION OF PILLAR-SPACING PLAN. (SEE ALSO FIG. 2.)

Pillars near open workings are large and up to 200 ft. apart. Pillars in next row are smaller but are only 125 ft. apart. First row from bottom is spaced 60 to 70 ft. between pillars and at present pillars are being left at 50-ft. intervals. Closer spacing of smaller pillars gives greater roof support.

These were 30 ft. square and spaced at about 175-ft. centers. Many of these crushed badly and the roof scaled into great arches between some pillars.

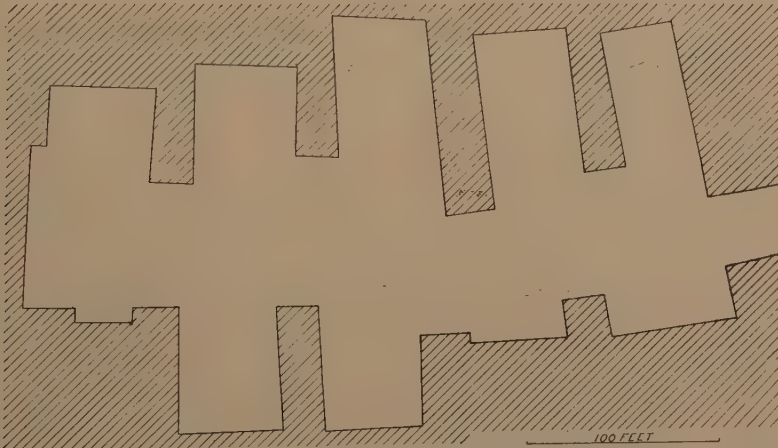


FIG. 13.—PLAN OF COVERED KITCHEN QUARRY, WEST RUTLAND, VT., SHOWING PILLAR PLAN IN COVERED, VERTICAL MARBLE DEPOSIT.

These attached or rib-type pillars are about 20 ft. thick and will be about 80 ft. high when the vertical part of the deposit is exhausted.

Present pillars are about 50 ft. high and 18 to 25 ft. square and never more than 60 ft. between centers. They do not show incipient failure and the spacing seems entirely adequate. The distance from floor to roof is 90 ft. in the Danby deposit and pillars are correspondingly stouter;

they are 20 to 30 ft. square and approximately 75 ft. between centers. Present spacing in both groups of quarries is equivalent to 50 ft. between supported roof sections.

Spacing in Vertical Deposits.—Pillars in vertical deposits, as at Pittsford Valley, West Blue and Covered Kitchen quarries, are not cut free from the side walls but are left as ribs (Fig. 13). The transverse distance from pillar to pillar is 65 to 75 ft. and longitudinally from rib to rib is 50 ft. The rib form gives the pillars greater rigidity and ability to support a jointed roof, a very important feature where the vertical height of the quarried space is great.

DISCUSSION

(*Oliver Bowles presiding*)

C. H. BEHRE, JR.,* Evanston, Ill.—Is there any tendency for marble to fracture during the quarrying process, possibly at right angles to the bedding plane?

G. W. BAIN.—Most stone at depth of 150 ft. or more has an inherent stress that tends to produce cracks parallel to the original bedrock surface. To overcome the effects of stress, two channel cuts are made and the fractured strip between is removed. The remainder of a bench or floor will be sound. The exact configuration of the stress breaks depends on the form of the bedrock surface.

O. BOWLES,† Washington, D. C.—It is significant that the rate of cutting marble with a wire saw is 3 to 4 sq. ft. an hour while the rate of cutting slate is 27 sq. ft. an hour. Why should this be?

G. W. BAIN.—The openings between grains of Vermont marble are only 1/10,000-in. wide; they are 10 to 15 times as wide in Carrara marble, which accounts for greater rapidity of sawing in Italian quarries; the individual grains of slate are torn away from one another much more easily than are the crystals of marble.

E. R. LILLEY,‡ New York, N. Y.—What is the proportion of waste in underground work? Will the pillars ever be recovered?

G. W. BAIN.—The pillars can never be recovered because if one were removed the excess stress on the others would cause cracks and would ruin the product. The waste in the quarry is about 40 per cent. About 60 per cent of the gross production is removed as quarry blocks. About 20 per cent of the gross production actually gets into a building. So far the *total* cost of underground work, where the deposit is as thick as at West Rutland or Danby, is about the same as in open quarries. However, this is due to the greater efficiency of large operations at these localities.

E. R. LILLEY.—Is there not more wastage in underground than in open-pit work because of the pillars that are left?

G. W. BAIN.—Yes, but the operators are philosophical on that phase of operations and most of them would say that there was no difference in wastage. They prefer to regard each 50-ft. face between pillars as a separate quarry.

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Geological Study of Gravel Concrete Aggregate of the Tennessee River

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(Washington, D. C., and College Park Meeting, October, 1937)

THIS study was undertaken primarily to determine the reasons for certain variations in the soundness of gravel aggregate taken from a number of widely separated points on the Tennessee River. Under laboratory tests, concrete made with these various aggregates in some cases exhibited marked differences in specific gravity, crushing strength and durability. Gravel was studied in detail from Chattanooga to the mouth of the river and to a lesser extent from Chattanooga to Knoxville. The causes of variation were determined, and variational curves were constructed, which accurately predict the composition of aggregate at any point on the river and therefore its relative suitability for use in concrete. The aggregate at any point is suited for making sound concrete; the variance of suitability is due to varying percentages of sandstone and weathered material, which are less sound than other constituents.

INTRODUCTION

When it is stated that the aggregate from one point is inferior to that from another, it must not be construed to mean that the gravel is unfit for use. The classification upon which such aggregate ratings are based is largely arbitrary, thus inferiority or superiority is merely relative.

Samples of gravel were selected from stockpiles at the dams of the Tennessee Valley Authority under construction, from the stockpiles of private companies along the river, and from samples taken by the gravel-exploration barge of the T. V. A. One hundred or more gravels of each size studied were selected at random by taking small handfuls while walking around the piles. These were separated according to the different mineral or rock constituents present, and the percentage of each constituent was obtained by counting. At least three such samples were taken from each size, and all checked with one another within a few per cent. The results were then averaged. In order to check on the accuracy of such sampling, a whole sample was taken from one point after the gravel was thoroughly mixed. The two methods gave approximately identical results.

SOURCES AND TYPES OF MATERIALS

The sources of gravel found in the Tennessee River are numerous and varied. The river itself traverses several geologic provinces, each

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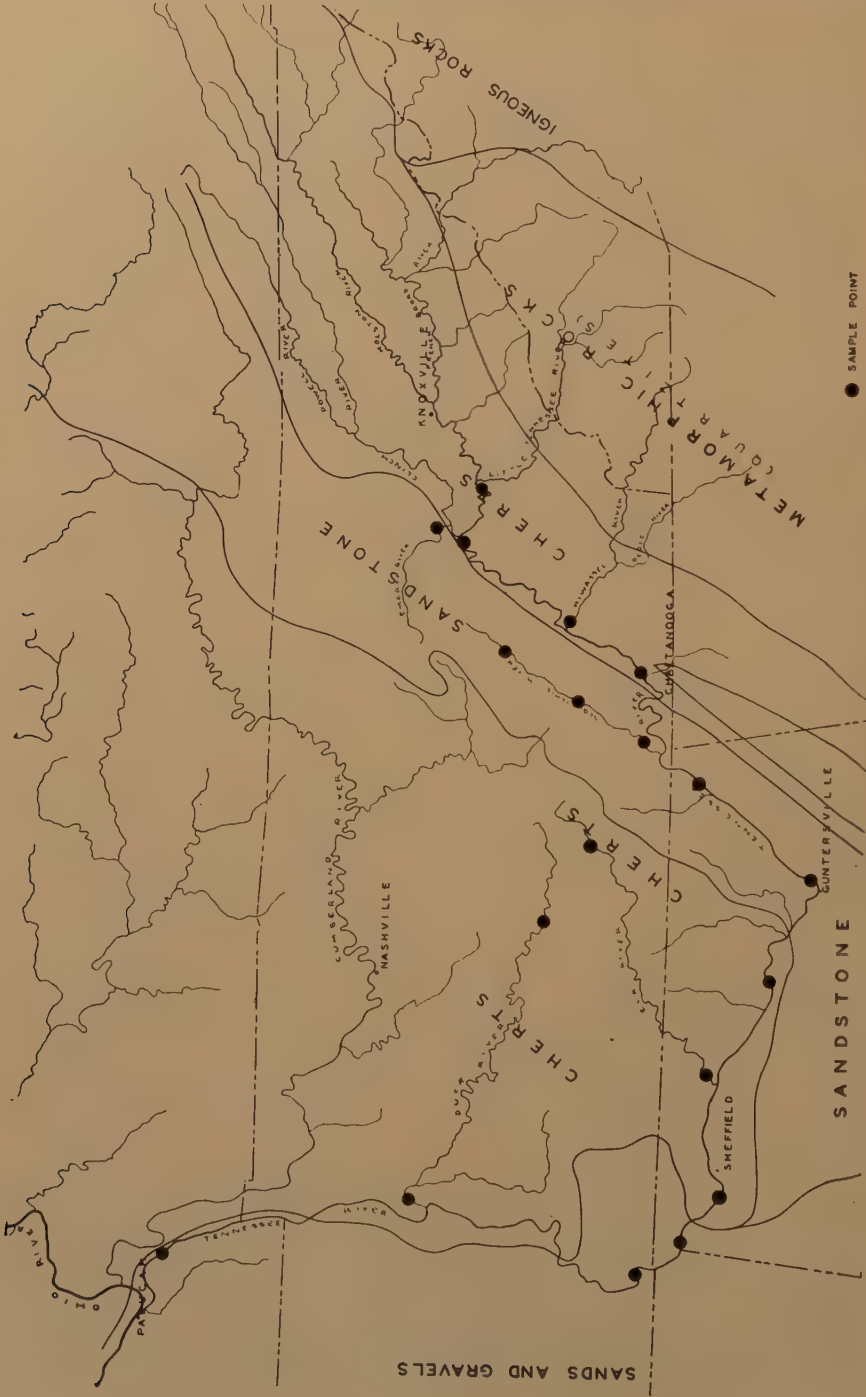


FIG. 1.—DIAGRAMMATIC GEOLOGIC MAP OF TENNESSEE VALLEY.

underlain by rocks peculiar to it, and still other types of rocks are drained by its tributaries. The principal sources of the most important rock types found in the gravel are shown on the accompanying geologic map (Fig. 1).

Chert.—Chert is a major constituent. In its original state it occurs as beds, lenses and nodules of noncrystalline silica embedded in the limestones of the region. Upon decay of the limestone, this insoluble material is set free in the form of angular blocks and masses of all sizes which, through wash and erosion, finally reach the streams to become rounded and subangular gravels. Owing to the cleavable character of most cherts, the breakdown of large masses into fragments of aggregate size is easily accomplished by splitting along these planes under natural weathering.

After the stream is reached other changes take place. Iron is precipitated from the water and, probably because of molecular attraction, forms a "skin" of brown iron oxide on the surface of the pebbles. Some chert, being primarily of a porous structure, suffers solution by weak natural acids and becomes porous and light.

There are several varieties of chert. That in the river in east Tennessee, and that brought in by the Sequatchie River in Alabama, is derived chiefly from the Knox dolomite and is porous, even vesicular in structure. Chert from the Bangor limestone along the river from Sequatchie Valley to Guntersville, Ala., and from Bear Creek in Alabama, is dark, heavy and dense. That from the Fort Payne formation in northern Alabama, which is also brought in by Elk River in Alabama and Duck River in west Tennessee, is similar to the chert of the Bangor, although it is lighter in color and not quite so dense. It tends also to maintain a blocky, angular shape even after long stream wear. The Warsaw and St. Louis limestones, the two chief contributors in northern Alabama and west Tennessee, produce a finely porous, massive chert.

Unsound Chert.—Unsound chert is sometimes abundant and is an undesirable element. The lack of soundness may be attributed to several causes. Chert may be naturally porous, like much of that from the Knox and Warsaw formations, or it may reach such a state under long weathering in the river. Because of this porosity, its specific gravity is lowered, and its absorption becomes too high. Some chert, even though dense and hard, may be so easily fractured and possess so many incipient fracture planes that it will split and disintegrate under freezing and thawing action.

The Tuscaloosa formation crops out in the vicinity of Bear Creek in Alabama and lines the Tennessee River for several miles upstream and downstream from the mouth of Bear Creek. It also occurs in the vicinity of Paducah, Ky. It is made up chiefly of chert gravels that have become

porous through long weathering. These are washed into the river and must be considered.

Quartzite.—Quartzite is a metamorphosed sandstone which, through heat and pressure, has become very hard, dense and durable. It is abundant in the river in its upper and middle portions, since it originates in western North Carolina and the eastern edge of Tennessee, being brought in by the Hiwassee, French Broad and Little Tennessee Rivers. It is always in the form of well rounded or discoidal pebbles and boulders. Apparently this is the most persistent of the gravels. It is found in the river from its point of entry to the mouth of the Tennessee and therefore must resist corrasion for great distances.

Sandstone.—Sandstone gravels reach the river from the Pennsylvanian sandstone areas of the Cumberland Plateau in Tennessee and northeast Alabama. The Emory River is a heavy contributor, but the numerous short streams descending rapidly from the escarpments bring down great quantities. In the main, the sandstone gravels are rather soft and friable. Because of this, they break down easily under river action and are not found in abundance very far from the areas in which they originate. A minor source of sandstone is the thin Chester formation of northern Alabama, gravel from which reaches the river chiefly by Bear Creek in the northwestern corner of the state.

Quartz.—Except in the fine sand of the river, quartz is a minor constituent of the aggregate. The Pennsylvanian sandstone areas and the breakdown of sandstone and quartzite gravels in the river have given an abundance of fine sand, which consists almost entirely of quartz. Quartz pebbles in the aggregate of larger sizes are present in only very small amounts and take the form of flattened smooth discoids. The origin of this is the crystalline belt of western North Carolina and extreme eastern Tennessee.

Minor Constituents.—Several rocks appear in amounts representing not more than one or two per cent. Igneous rocks consisting of granite, pegmatite and other rocks are brought in from North Carolina. In the larger gravels occasional pieces of limestone are found.

RELATIVE SUITABILITY OF CONSTITUENTS FOR CONCRETE AGGREGATE

Upon the basis of impermeability, hardness, durability and resistance to weathering, the following principal constituents of the gravel are named in the order of decreasing fitness for use in concrete: (1) quartzite, (2) sound chert, (3) weathered chert, (4) sandstone.

A high percentage of quartzite is greatly to be desired because it fills all requirements for sound aggregate. Sound chert is also an excellent concrete material provided it is not easily fractured and broken.

Weathered chert, and both weathered and sound sandstone have a deleterious effect upon aggregate. The chert, being porous, absorbs water, and under freezing and thawing action breaks down rapidly. Sandstone, particularly weathered sandstone, is also very porous, and in addition the grains are loosely cemented together so that the rock easily

TABLE 1.—*Sodium Sulphate Soundness Test*
(ASTM Designation C 89-35T)
5 and 10 Cycles

Laboratory No.	Identification	Weight of Test Fractions before Test, Grams (Bone Dry)	Cycle No.	Loss in Weight Due to Disintegration, Per Cent	Remarks
23600	¾-in. chert	933	5	3.0	Shows sign of failure at first cycle. No spalling.
23600	¾-in. chert	933	10	12.1	
23601	1½-in. chert	689	5	13.2	Shows sign of failure at first cycle. No spalling.
23601	1½-in. chert	689	10	27.6	
23602	¾-in. sandstone	862	5	31.5	Shows sign of failure at first cycle. No spalling.
23602	¾-in. sandstone	862	10	56.8	
23603	1½-in. sandstone	903	5	53.5	Shows sign of failure at first cycle. No spalling.
23603	1½-in. sandstone	903	10	67.3	

TABLE 2.—*Freezing and Thawing Tests^a*
10 and 25 Cycle Reports

Laboratory No.	Identification	Weight of Test Fractions before Test, Grams (Bone Dry)	Cycle No.	Loss in Weight Due to Disintegration, ^b Per Cent
23600	¾-in. chert	911	10	1.76
23600	¾-in. chert	911	25	2.63
23601	1½-in. chert	790	10	1.14
23601	1½-in. chert	790	25	2.03
23602	¾-in. sandstone	931	10	4.08
23602	¾-in. sandstone	931	25	7.1
23603	1½-in. sandstone	877	10	9.24
23603	1½-in. sandstone	877	25	12.0

^a The specimens were first saturated in water, then frozen half submerged in water to a temperature of 0° F., after which they were thawed at 80° F.

^b Disintegration is classified as material passing a number 4 sieve.

disintegrates. A comparison of the resistance of sandstone and weathered chert to artificial freezing and thawing and to sodium sulphate soundness

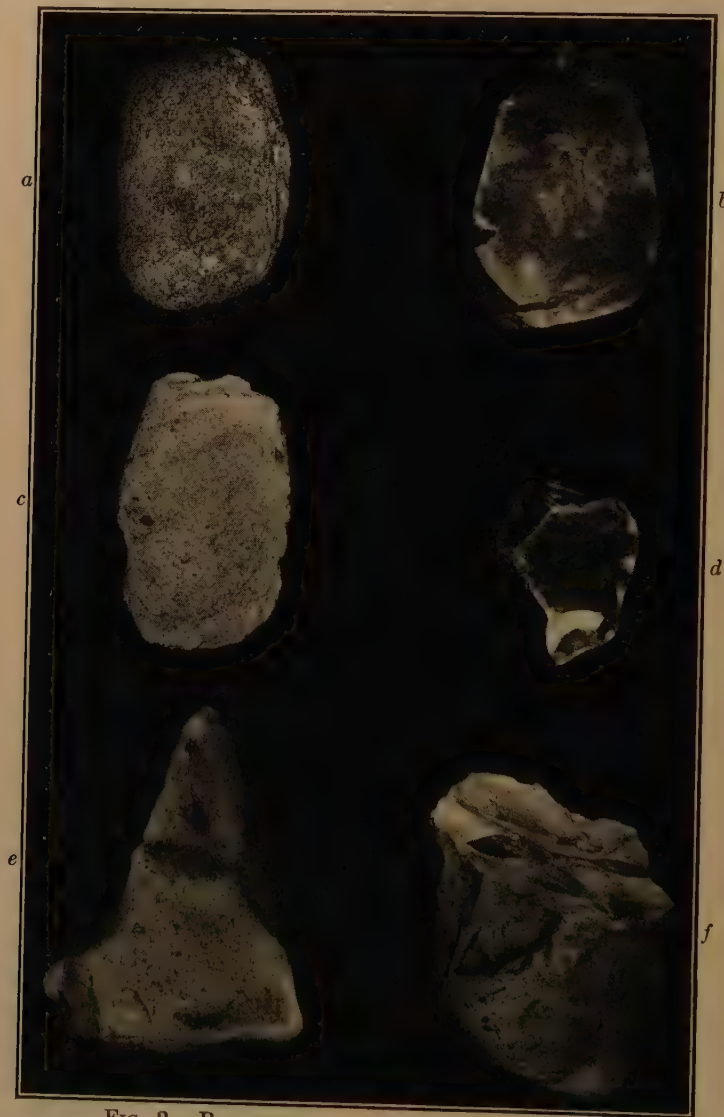


FIG. 2.—ROCK TYPES FOUND IN 3-INCH GRAVEL.

- a.* Porous, fossiliferous chert from Warsaw formation.
- b.* Dense, black chert from Bangor formation.
- c.* Badly weathered chert, which was porous in its original state.
- d.* Common form assumed by dense chert owing to repeated splittings.
- e.* Chert that has become finely porous through weathering.
- f.* Light colored, dense chert from Fort Payne formation, showing fracture planes.

tests is brought out in Tables 1 and 2. The samples were selected from the stock piles at Guntersville Dam. From the comparison of these

tests, the five-cycle sodium sulphate test appears to be more severe than the 25-cycle freezing and thawing tests.

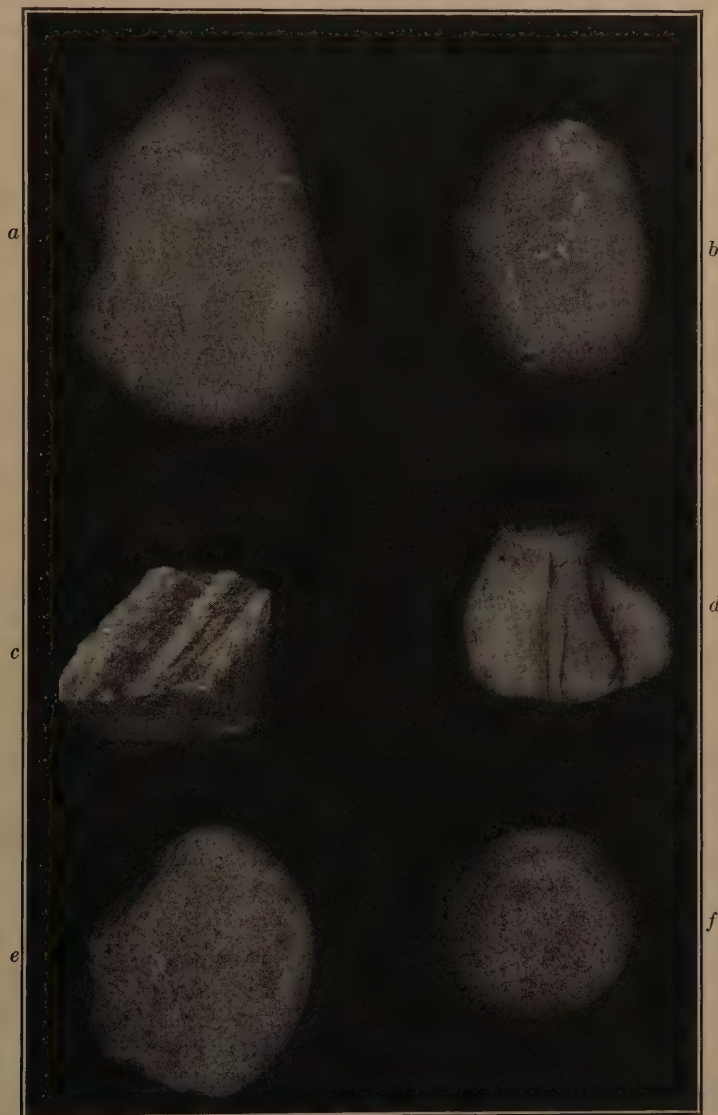


FIG. 3.—ROCK TYPES FOUND IN 3-INCH GRAVEL.

- a, b.* Dense, smooth gravels of quartzite.
- c.* Crushed gravel of quartzite showing angular form of breakage.
- d.* Vein quartz.
- e, f.* Sandstone gravels; *e* is in process of disintegration.

Although the weathered chert disintegrates, it does not do so to a dangerous extent, but the sandstone is extremely weak in its resistance

to freezing and thawing, and therefore represents the least desired constituent. Its presence in large quantities in the face of a concrete structure may easily cause spalling of the concrete under freezing and thawing or other weathering agencies. The percentage of soft sandstone in an aggregate should be carefully determined.

VARIATIONS IN AGGREGATE AND THEIR CAUSES

Variation Due to Weathering Properties

In an aggregate sample taken from the river at any one point, there are variations in composition dependent upon the gravel size of the material examined; for example, at Chickamauga the 3-in. aggregate contains 15 per cent sound chert; the $1\frac{1}{2}$ -in., 22.2 per cent; the $\frac{3}{4}$ -in., 42 per cent and the $\frac{3}{8}$ -in., 51.5 per cent. The fine sand shows practically no chert.

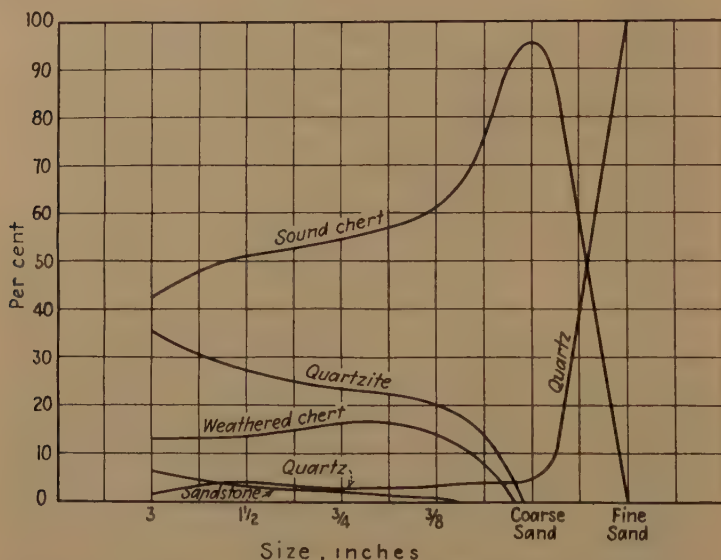


FIG. 4.—AGGREGATE VARIATION BY SIZE.

Theoretically the composition of the $\frac{3}{8}$ -in. aggregate should be the same as that of the 3-in., but owing to differences in resistance to river action and weathering, there is often a wide variation between the sizes. The graph of Fig. 4 is an average of aggregate compositions for the entire river and exemplifies the changes in composition of the entire aggregate dependent upon the size taken.

An explanation of these changes may best be brought out by describing the manner of decomposition of each rock type.

Chert.—Large pieces of chert, although quite hard, tend to break up easily into smaller pieces because of the numerous and closely spaced cleavage planes. Thus we find sound chert increasing from 42 per cent

in the 3-in. size to 61 per cent in the $\frac{3}{8}$ -in. gravel. Here, other constituents of the aggregate of this size begin to disintegrate completely and sound chert climbs rapidly to 95 per cent in the coarse sand, the next smaller size. Chert apparently cannot persist beyond the size of coarse sand, however; because of its small surface area and brittleness, it is quickly pounded into silt by larger gravels. Thus we find less than 1 per cent in the fine sand. Weathered chert does not persist beyond the $\frac{3}{8}$ -in. size.

Quartzite.—Quartzite breaks down slowly and evenly until $\frac{3}{8}$ in. is reached. The process of disintegration is chiefly one of mechanical attrition whereby it is chipped and worn through contact with other pebbles, or through sliding along the bottom. From 35 per cent in the 3-in. aggregate, quartzite falls to 20 per cent in the $\frac{3}{8}$ -in. There is no tendency toward multiplication by splitting and cleaving, as in the chert. Three-eighths inch is again a critical point of change in the aggregate for the small quartzite pebbles; having been weakened by chemical action as well as mechanical, they are no longer able to resist the grinding action of the larger gravels, and quartzite disappears into fine sand.

Sandstone.—As shown by the curve in Fig. 4, sandstone is very weak. From 7 per cent in the 3-in. material, only a bare 1 per cent is found in $\frac{3}{8}$ -in. gravel, after which it is represented only by fine sand.

Quartz.—Quartz in the larger aggregate sizes is relatively rare and unimportant. The 3-in. aggregate contains approximately 2 per cent, which increases to 5 per cent in the $1\frac{1}{2}$ -in. gravel, owing probably to the splitting up of the larger fragments, after which it varies little until the coarse sand is reached. It is beyond this point, in the fine sand of the aggregate, that quartz assumes the dominant position. Quartz sand is the end product of the disintegration of quartzite and sandstone, and is the only material that can exist in this small size. Thus, discounting a fraction of 1 per cent of mica and ferromagnesian minerals, the fine sand of the Tennessee River is all quartz.

Variation of Aggregate in the Tennessee River and Description of Aggregate Profiles

The important variations of aggregate are those between gravel samples taken at separated points on the river, and the chief value and contributions of this report are the accompanying variational curves that make up an "aggregate profile" of the Tennessee River from Chickamauga Dam to Gilbertsville, Ky. By the use of these curves, the composition of the aggregate at any point in the river can be determined correctly. The percentage figures from which these curves were constructed are arranged in Table 3.

The primary causes of variation along the river are: (1) the types of rock underlying the areas traversed by the river, (2) large tributaries

TABLE 3.—*Percentage Figures*

Sandstone	Quartzite	Quartz	Chert		Place
			Sound	Weathered	
3-INCH AGGREGATE					
	2.4		64.0	33.6	Gilbertsville
	6.9	0.5	68.6	24.0	Pickwick
4.0	15.5	1.0	66.5	13.0	Riverton
1.0	15.0	1.0	68.0	15.0	Pride Landing
4.6	51.8	1.0	33.4	9.2	Triana
21.5	50.5	2.5	18.5	7.0	Guntersville
15.5	71.8	2.0	8.5	2.2	Widows Bar
6.6	70.2	7.0	15.0	1.2	Chickamauga
53.2	284.1	15.0	342.5	105.1	
6.7	35.5	1.9	42.7	13.2	
1½-INCH AGGREGATE					
			76.8	23.2	Gilbertsville
	2.5	1.0	72.0	24.5	Pickwick
1.0	4.0	3.0	81.5	10.5	Riverton
1.5	2.0	3.0	68.0	26.5	Pride Landing
1.0	43.0	2.0	48.0	6.0	Triana
13.5	55.5	10.0	11.5	9.5	Guntersville
9.0	52.0	4.0	29.0	6.0	Widows Bar
2.9	62.8	9.1	22.2	3.0	Chickamauga
28.9	221.8	32.1	409.0	109.2	
3.6	27.7	4.0	51.1	13.6	
¾-INCH AGGREGATE					
			54.0	46.0	Gilbertsville
	3.2	1.0	73.0	23.0	Pickwick
	4.2	1.7	81.6	12.5	Riverton
	3.5	2.5	71.2	22.8	Pride Landing
1.0	43.0	4.0	50.0	2.0	Triana
10.0	36.5	5.0	32.5	15.0	Guntersville
3.3	54.5	4.8	33.0	4.2	Widows Bar
4.0	41.6	4.0	42.6	8.0	Chickamauga
18.3	186.5	23.0	437.9	133.6	
2.3	23.3	2.9	54.7	16.8	
⅝-INCH AGGREGATE					
	2.8	1.5	61.2	34.5	Gilbertsville
	11.2	4.4	64.2	20.2	Pickwick
	5.5	5.0	79.2	10.3	Riverton
	13.0	3.0	68.0	16.0	Pride Landing
	35.0	3.3	59.3	2.4	Triana
1.7	35.0	4.2	47.5	11.5	Guntersville
4.2	27.0	4.0	58.0	6.8	Widows Bar
2.0	33.0	4.5	51.5	9.0	Chickamauga
7.9	162.5	29.4	488.7	110.7	
1.0	20.3	3.7	61.1	13.9	

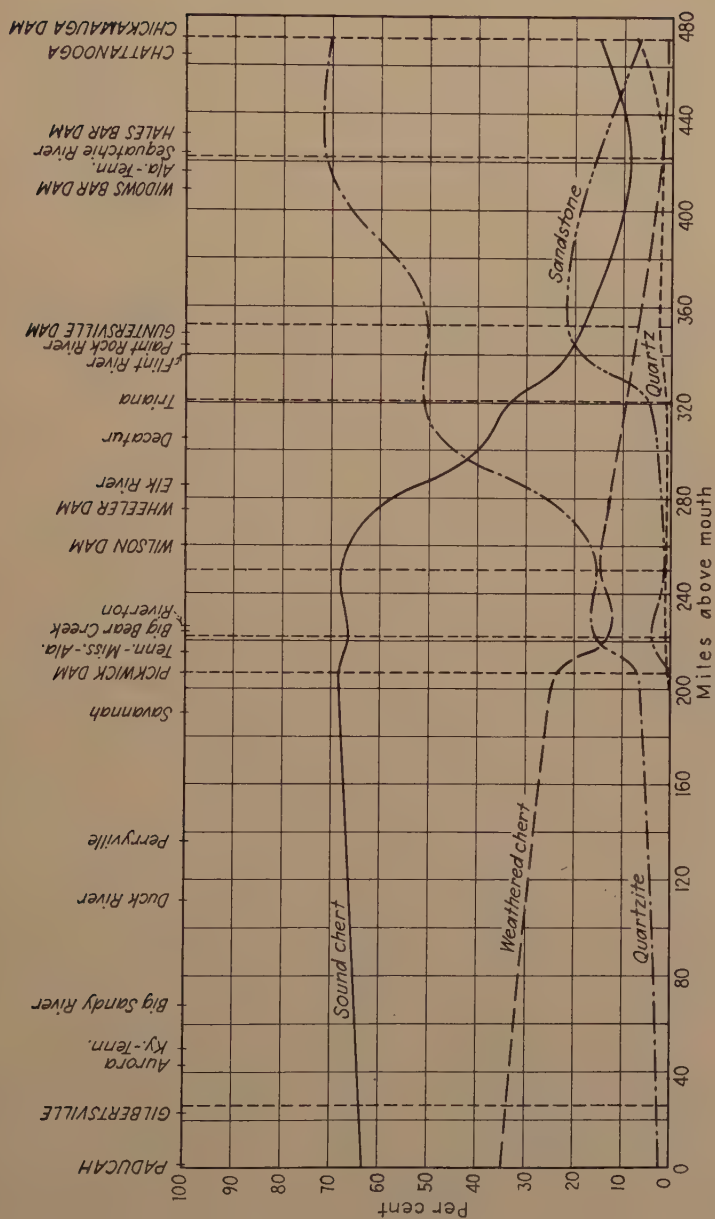


FIG. 5.—VARIATION OF 3-INCH AGGREGATE IN TENNESSEE RIVER.
Vertical dotted lines represent sample points.

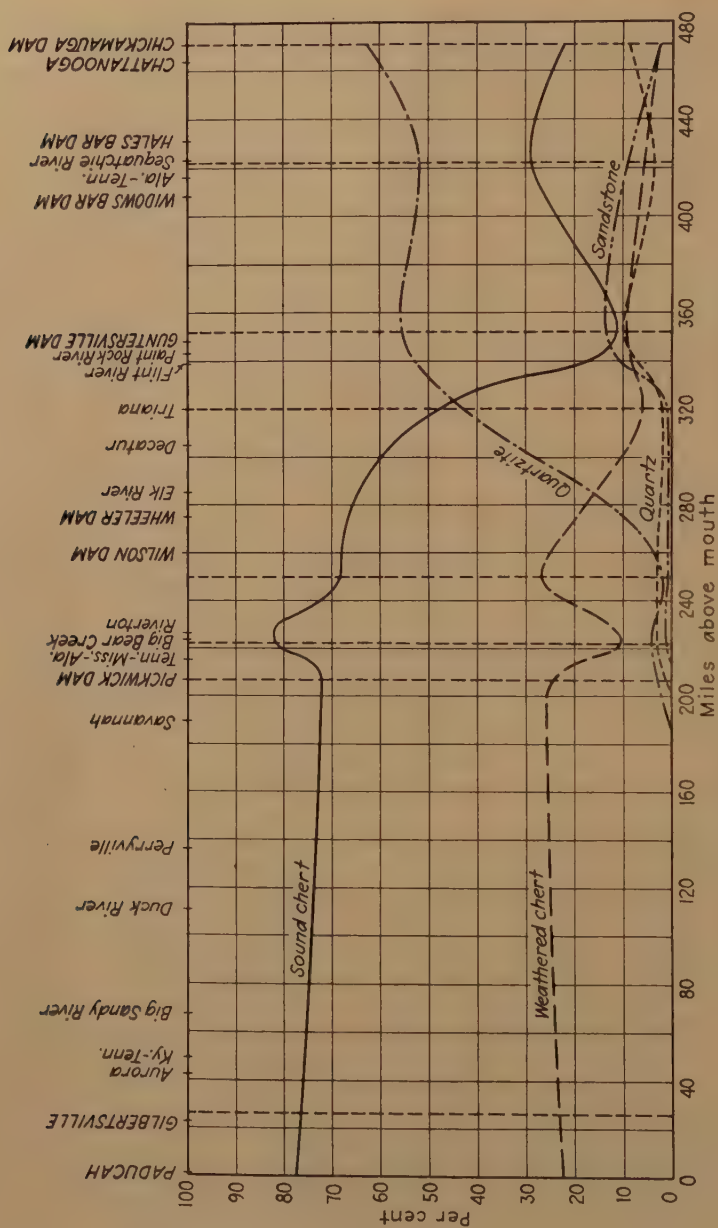


Fig. 6.—VARIATION OF 1½-INCH AGGREGATE IN TENNESSEE RIVER.
Vertical dotted lines represent sample points.

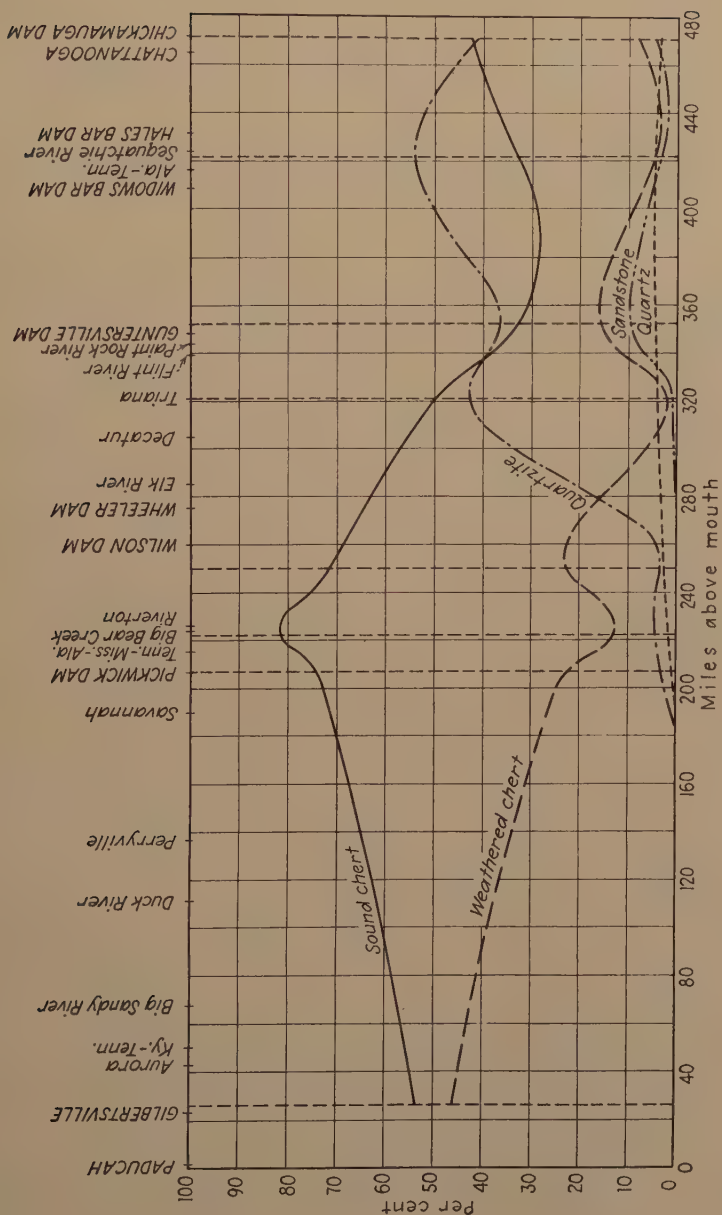


FIG. 7.—VARIATION OF $\frac{3}{4}$ -INCH AGGREGATE IN TENNESSEE RIVER.
Vertical dotted lines represent sample points.

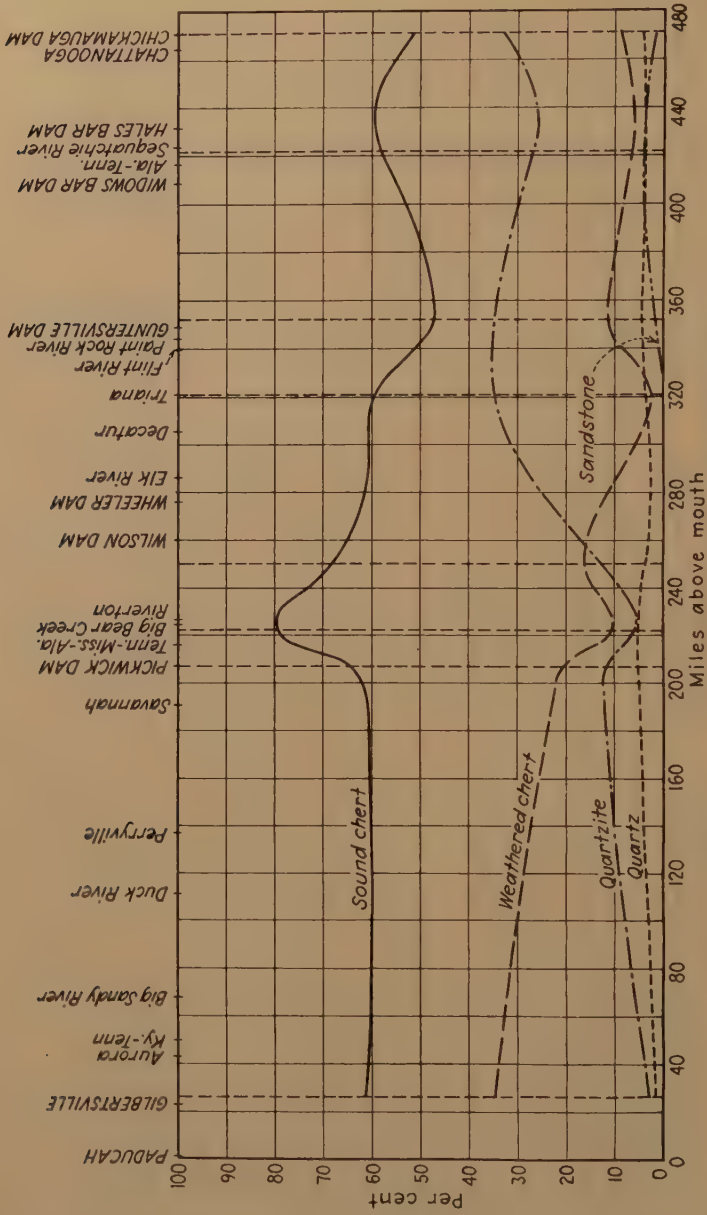


FIG. 8.—VARIATION OF $\frac{3}{8}$ -INCH AGGREGATE IN TENNESSEE RIVER.
Vertical dotted lines represent sample points.

bringing in new material, and (3) mechanical and chemical attrition of gravel as it is moved downstream. The composition and condition of the gravel at certain significant points along these profiles are as follows:

3-inch Aggregate (Fig. 5).—At Chickamauga Dam, quartzite is the predominant constituent because the Hiwassee River upstream has contributed a load of gravel, consisting almost entirely of this rock, increasing the amount already carried in the river. Chert and sandstone are low in proportion, aggregating no more than 30 per cent, and any aggregate taken from this vicinity, or from points between Chickamauga and Widows Bar should be excellent. After leaving Chattanooga, the Tennessee River flows through the sandstones of the Cumberland Plateau until it reaches Guntersville. Large amounts of sandstone and some sound chert are acquired in this passage, the gravel's content of sandstone reaching its peak a few miles above Guntersville Dam. Although the percentage of quartzite is still high, the presence of considerable sandstone and weathered chert causes the aggregate to be quite inferior.

Turning westward from Guntersville, the river crosses the cherty limestones of northern Alabama, and sound chert is derived from the Fort Payne and Bangor limestones, the content reaching a maximum between Wilson and Pickwick Dams. The excellent character of the 3-in. aggregate at Triana, a few miles above Decatur, should be noted, the sound chert and quartzite percentage being high.

A small fluctuation in the curves may be noted at Big Bear Creek. Apparently no large gravels of unweathered chert are carried by Big Bear, but a small quantity of sandstone from the south makes its entrance through this stream. From Bear Creek to its mouth, the river passes through rocks that yield only chert and some sand. Thus, the percentage of weathered chert rises slowly and evenly because of chemical action upon it by the river water. Sound chert decreases in proportion, and quartzite, the only other rock to persist the entire length of the river, falls to 2 per cent.

1½-inch Size (Fig. 6).—Profiles for the 1½-in. gravel behave in much the same way as those for the 3-in. aggregate, with two exceptions: (1) quartzite does not exist in this size beyond Pickwick Dam, and (2) fresh material from Bear Creek raises the sound chert percentage to 82. The sound chert is derived from the Bangor limestone belt through which Bear Creek flows. Oddly enough, the sound chert increases in amount from Bear Creek to Gilbertsville, owing probably to the splitting up of larger pieces of the dense chert. In addition, in west Tennessee, Duck River contributes a considerable amount of sound chert of this particular size.

¾-inch Size (Fig. 7).—The behavior of this size from Chickamauga to Bear Creek is similar to that of the 3 and 1½-in. gravels. From Bear Creek to the Ohio River, the content of weathered chert rises rather

sharply, reaching 46 per cent at Gilbertsville, a maximum for any size at any point on the river.

$\frac{3}{8}$ -inch Size (Fig. 8).—The outstanding characteristic of the curves for this size is the persistence of 2.8 per cent quartzite to Gilbertsville. Other than that, the profile does not differ markedly from those of other sizes. Some of the old gravel terraces lining the course of the river in west Tennessee are made up of very small quartzite pebbles, which probably are responsible for the 2.8 per cent at Gilbertsville.

Aggregate Variation between Chickamauga and Knoxville.—Hand-picked samples from this reach of the river, therefore not truly representative, were examined. One sample was obtained from a point just above the mouth of the Emory River, near Kingston, Tenn., which contained a very high percentage of quartzite. It is believed that aggregate from this part of the river is superior to that discussed in this report because of the absence of sandstone and weathered chert.

APPLICATION OF STUDIES

It should be emphasized that the classification of the various constituents of the gravel is not a strictly geologic one, but rather is based upon standards of soundness and suitability for use as concrete aggregate. The culmination of the studies, of course, is the compilation of the profiles described. To break them down into figures representing percentages of sound and unsound material, Table 4 has been prepared:

TABLE 4.—*Percentage of Sound and Unsound Material*

	By Size			
	3-inch	1½-inch	¾-inch	$\frac{3}{8}$ inch
Per cent—sound concrete material.....	80.1	82.8	80.9	85.1
Per cent—unsound concrete material.....	19.9	17.2	19.1	14.9

AT SAMPLE POINTS

	Sound	Unsound
Gilbertsville.....	65.7	34.3
Pickwick.....	77.1	22.9
Riverton.....	87.2	12.8
Pride Landing.....	79.3	20.7
Triana.....	93.4	6.6
Guntersville.....	77.6	22.4
Widows Bar.....	87.2	12.8
Chickamauga.....	90.3	9.7

The figures largely speak for themselves. At Guntersville, however, it should be noted that more than half of the 22.4 per cent unsound material consists of weathered sandstone. With the exception of that from

Gilbertsville, the aggregate at Guntersville is probably inferior to that from any other portion of the river, simply because of the high content of sandstone. It is well suited, however, for making sound concrete.

DISCUSSION

(*Oliver Bowles presiding*)

S. WALKER,* Washington, D. C.—Do you feel that the sodium sulphate test and the freezing and thawing test accurately differentiate between sound and unsound chert?

E. L. SPAIN.—I think freezing and thawing amply suit the needs of concrete aggregate in our particular region and would probably determine sound and unsound aggregate in more northern regions. I think the sodium sulphate test is too severe, because constantly putting the aggregate back into the solution causes crystalline growth in the pores. Every time it is put back, more liquid enters the pores and forms a supersaturated solution, and that constant crystalline growth I think is much more extreme than the freezing action. I believe that freezing and thawing constitute a fair test of durability. It has been our experience that they do. Those tests, I might add, are adapted after the tests used by the Pittsburgh Testing Laboratories. The man in charge of that work was associated with them for a number of years.

S. WALKER.—Cherts are common constituents in gravels in certain localities, and it has been a matter of great interest to us to endeavor to determine what is the dividing line between unsound cherts and sound cherts. I must confess that we have not found any entirely satisfactory solution to that question. Of course cherts have a rather high thermal coefficient of expansion as compared with other aggregates, and I wonder if some of the breaking down that Mr. Spain found in the freezing and thawing tests may not have been due to the accompanying variation in temperature and if, in a structure with temperature changes such as might be encountered in Tennessee, the same breaking down would have occurred.

The author mentioned only the porosity of the sandstones. It is true that a great many sandstones, perhaps the larger proportion of them, have a high degree of porosity, but it has been my experience that some of the more porous have produced concrete of a high degree of durability. I have in mind, in particular, a sandstone in Minnesota, which has an absorption of some 7 to 9 per cent. Concretes made with that material have shown a very high degree of durability, when properly made. Is there not some other property of the sandstone, such as the nature of the binding material, that has the most important effect on its resistance to weathering?

E. L. SPAIN.—The binding material is silica, for the most part. There is one other mechanical feature, which I did not bring out. I have noticed this at Guntersville dam. We were bringing in plus 3-in. aggregate containing a high percentage of sandstone, and after it was unloaded over the barges by clamshells and run up to the aggregate tower and dropped on the piles, about 70 per cent of the plus 3-in. aggregate would go through a 2-in. screen and therefore ruin the sizing. That, of course, is a mechanical feature, but it illustrates how loose the cement is.

E. F. BURCHARD,† Washington, D. C.—Have they made any crushing tests of the concrete made from these aggregates?

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† Mining Geologist, U. S. Geological Survey.

E. L. SPAIN.—In some of the crushing tests, we found that the breakage of cylinders was caused by a sandstone boulder or cleavable chert boulder or pebble. Of course, in the mass aggregate that would not amount to anything because the pieces are well separated from one another. About the only effect would be, down in our country, the spalling of concrete. I have noticed it in a number of our bridges, especially in gravel made from the low regions of the river, where the weathered chert content is rather high.

G. W. WARD,* Washington, D. C.—Have you frozen and thawed concrete containing chert aggregate?

E. L. SPAIN.—Yes.

G. W. WARD.—My experience has been, in one case in particular with chert from the Ohio River, that the chert itself, not in concrete, frozen and thawed from 150 cycles showed practically no failure. It was a dense, hard, compact chert. One hundred fifty cycles of the sodium sulphate soundness test showed practically no failure. That same chert, or chert from that same sample, placed in concrete and frozen and thawed for 35 cycles entirely disrupted the concrete. Is that your experience?

E. L. SPAIN.—Was that not because the moisture entered the bond between the aggregate and the concrete around it, the cement around it therefore splitting away from the pieces of aggregate, rather than splitting the aggregate itself?

G. W. WARD.—No; the aggregate itself was definitely shattered.

E. L. SPAIN.—No, I do not know that I have seen any instance of that. I have noticed the bond splitting apart but I do not know of any instances of the aggregate itself doing that.

E. F. BURCHARD.—Were these gravels all from the river bed, dredged gravels, or were studies made on any bank gravels?

E. L. SPAIN.—They were all taken from the river bed itself, from gravel bars in the river.

A. T. GOLDBECK,† Washington, D. C.—We have been trying for some years to correlate the various kinds of laboratory tests for soundness with the probable effect of particular aggregates in concrete, and we have run into all sorts of inconsistencies. It is rather important to take into account the fact that aggregates, when they are unsound or when they appear to be unsound in these accelerated tests, fail in different ways. For instance, there are aggregates that fail in a sort of explosive way. They expand to a very considerable extent, often into smaller particles, which appear to be sound. On the other hand there are certain materials that disintegrate into rather finely divided particles. The question arises as to what will be the effect of those different types of aggregate when they are put into concrete. We have had experience with certain aggregates that show a very high percentage of failure in both the sodium sulphate test and freezing and thawing test, yet those same aggregates have been used in rather important structures, structures that have been subjected to water action and to freezing and thawing for many years yet nothing seems to be happening. So I still feel that we have much to learn about the proper correlation of the various accelerated tests with the behavior of concrete in service. We encounter various inconsistencies in the behavior of the sodium sulphate test as compared with the

* Portland Cement Association Fellowship.

† National Crushed Stone Association.

freezing and thawing test. We are still struggling with the problem and we feel that we do not know just exactly how to interpret these various accelerated tests for soundness.

E. L. SPAIN.—That is true. This particular study was more or less an effort to explain those results in a general way, more than anything else. We did not go into detailed laboratory work on any part of it.

P. M. TYLER,* Washington, D. C.—I am not very familiar with the literature on sand and gravel. Is not this a novel method of making a geographical analysis of a region? I have never seen an areal analysis made in this way—making regular sample maps over a great territory. Is there novelty in the presentation of it?

E. L. SPAIN.—I think so. I have never seen it done before. We had the geology already worked out, and having worked in that area for a number of years we knew the type of rock to be derived from various areas, and it really was not very difficult to correlate those areas with what we found in the river.

S. WALKER.—Mr. Spain has described a novel method of charting the characteristics of materials from a wide area and I think that the information that has been developed will be an extremely valuable contribution to the literature and particularly for those who are interested in the areas that he charted.

I want to comment on one statement made by Dr. Ward, with respect to finding failures in freezing and thawing tests of concretes made with aggregates that had shown good resistance in both the freezing and thawing test and the sodium sulphate test. I agree with his statement but I do not want to run the risk of leaving the impression that all chert aggregates react in that way. It is a most puzzling problem. It is one which has worried us very much in a dollars and cents way for a great many years. We do know from experience that we have some very important sources of supply of gravels which consist of 75 to 90 per cent of chert which have been used with excellent success for a great many years in structures exposed to severe weathering conditions but, on the other hand, as interested as we are in that problem, we have not been able to draw any sharp line of demarcation between those cherts which cause us trouble and reflect discredit on the industry, and those which will stand up satisfactorily.

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Utilization of Slag in the Birmingham District, Alabama

BY JAMES R. CUDWORTH, MEMBER,* AND JOSEPH C. MEAD,† STUDENT ASSOCIATE
A.I.M.E.

(New York Meeting, February, 1937)

THE Birmingham district of Alabama has utilized the slag from its blast furnaces consistently since the earliest development of the slag industry. Today there are producers of slag cement who started business in 1900, and who have, under a progressive policy, expanded the number of their products to meet present demands. There are producers of slag aggregates and other crushed slag products who have been successful for periods of over thirty years. There are at the present two companies, Cheney Lime and Cement Co. and the Southern Cement Co., manufacturing puzzolan cement; two companies, Birmingham Slag Co. and Woodstock Slag Co., making slag aggregates and by-products; one company, Tennessee Coal, Iron and Railway Co., a subsidiary of the U. S. Steel Corporation, producing ground open-hearth basic slag for soil conditioning. Two companies, the Sloss Sheffield Steel and Iron Co. and the Woodstock Slag Co., make slag concrete in ready-mix plants and one company, the Superock Company, manufactures fine and coarse slag aggregates under a new granulation process. No Portland cement is made with slag as an ingredient, because dolomite is used so extensively as a blast-furnace flux that the magnesium content of the slag is excessive.

SOURCES OF SLAG

In the Birmingham district there are 22 blast furnaces, of which the majority are in blast. The present production is 165,000 tons of pig iron monthly; the production in 1935 was 1,297,960 gross tons¹. There are also two blast furnaces at Gadsden and one ferroalloy electric furnace at Anniston. From these two sources is obtained slag to approximately 1.1 tons per ton of pig iron.

HISTORY OF DEVELOPMENT OF THE INDUSTRY

The history of the local utilization of slag as an industry probably starts with the Birmingham Slag Cement Co. This company was

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¹ References are at the end of the paper.

merged with the Southern Cement Co. in 1901. In the slag-cement field the start of the Birmingham plants followed soon after the establishment of the first slag-cement plant in the country at Sparrow's Point, Md. Beginning with a plant having a capacity of 50 bbl. a day, the Southern Cement Co. has increased the plant to the present capacity of 2500 bbl. daily. Making initially one product, the Southern Cement Co. now produces three brands: (1) Magnolia stainless cement, (2) Magnolia Mason's Mix, (3) Magnolia mortar cement. The Cheney Lime and Cement Co. has produced for years a brand of slag cement marketed as Vesuvius cement. The use of slag cement mixed with Portland cement in concrete work has increased in recent years in the South. The history of the development of the industry for the production of slag aggregates and by-products is more difficult to trace. The first red hematite ore was smelted near Irondale on the outskirts of Birmingham in 1864². In 1876, the successful use of coke in making pig iron inaugurated the great industrial activity in the districts, and the utilization of the slag followed in due course. For many years slag has been used in Alabama for road building and paving, railway ballast and for concrete construction. More recently it has been used for concrete products. The three principal producers of crushed and screened slag at Birmingham have a combined annual producing capacity at the present time of approximately 3,000,000 tons. Most of these companies have erected ready-mix concrete plants in association with their crushing plants.

RAW MATERIALS

Most of the slag produced in the district is blast-furnace slag, which is a by-product of iron ore, dolomite, and coke in the production of pig iron; all of these are from Alabama ores except some iron sinter from Tennessee.

The red or hematite ore occurs in several beds or seams, in the Red Mountain, or Rockwood formation, of which the Big seam is most important commercially. The mineralogical composition of the Big seam varies somewhat at various places in the district and at different horizons in the seam. The principal constituents are red hematite mixed with silica, alumina and lime carbonate with the iron content remarkably constant. Typical analyses of the ore from the Big seam are given in Table 1³.

The ore from the Wenonah mine is practically self-fluxing and is typical of the character of the ore in a number of mines in this area.

The red ore (hematite) accounts for most of the iron ore used in the district. However, some brown ore is used, usually from irregular masses of concretionary form associated with clays. A representative analysis of this ore is as follows³: Fe, 50.39 per cent; SiO₂, 13.02; Al₂O₃, 2.41; Mn, 0.82; P, 0.32; H₂O, 5.38.

The iron sinter used in the Birmingham district is largely obtained from the Copper Hill-Ducktown district of Tennessee and is a by-product

TABLE 1.—*Typical Analyses of Ore from Big Seam*³

	Ishkoda Mine No. 13, Per Cent	Wenonah Mine No. 8, Per Cent	Muscoda Mine No. 4, Per Cent
Moisture.....	1.00	1.00	1.00
Iron (Fe).....	36.82	37.10	35.28
Silica (SiO ₂).....	17.90	13.08	10.38
Alumina (Al ₂ O ₃).....	3.03	3.05	3.12
Lime (CaO).....	13.67	16.02	19.35
Manganese.....	0.17	0.16	0.16
Phosphorus.....	0.37	0.36	0.30

of the sulphuric acid industry of that place. Some sinter is obtained from flue dust in the Birmingham district.

The fluxes used in the production of pig iron are mainly dolomite and limestone. The dolomite, which has largely replaced the limestone in use, occurs in the Knox dolomite formation. A typical analysis of the dolomite as quarried for flux is⁴: SiO₂, 0.70 per cent; Al₂O₃, 0.63; CaCO₃, 56.41; MgCO₃, 43.00.

Limestone, used principally in the open-hearth furnaces, is obtained from different localities, of which there are many in Alabama. A typical analysis of high-grade fluxing limestone is⁵: SiO₂, 1.64 per cent; FeO, Fe₂O₃, Al₂O₃, 0.50; CaCO₃, 96.60; MgCO₃, 1.22.

The coke used in the Birmingham district is obtained from several seams of coking coal, of which the Pratt and Mary Lee are the most important producers. Average analyses of coal from these two seams are given in Table 2.

TABLE 2.—*Analyses of Coal Used in Birmingham District*²

	Mary Lee Seam, Per Cent	Pratt Seam, Per Cent
Moisture.....	2.71	2.41
Volatile matter.....	25.61	27.77
Fixed carbon.....	60.77	61.07
Ash.....	10.92	8.74
Sulphur.....	1.50	1.82

Fluorspar is used as a flux in the basic open-hearth furnaces to increase the fluidity of the slag and to aid in desulphurizing the steel. For this purpose most of the fluorspar used in the Birmingham district comes from Kentucky and Illinois. The standard specifications⁵ call for a minimum

of 85 per cent CaF_2 and a maximum of 5 per cent SiO_2 . Most of the fluorspar appears in the slag.

KINDS OF SLAGS

The blast-furnace slag varies somewhat in chemical composition, depending upon the type of pig iron that is being run, but for any one furnace it is remarkably uniform. Representative analyses of the slag⁶ are given in Table 3. The final slags from the basic open-hearth furnaces are higher in bases than the blast-furnace slags. Representative analyses are given in Table 3.

TABLE 3.—*Analyses of Slag*

	Foundry Pig, Per Cent ⁶	Basic Pig, Per Cent ⁶	Final Slags from Basic Open- hearth Furnaces, Per Cent ⁷
Silica.....	36.90	34.55	20.74
Alumina.....	12.36	11.34	3.55
Phosphorus.....			2.85
Lime.....	44.10	45.60	40.90
Magnesia.....	4.52	6.21	9.67
Iron oxides.....			16.08
Ferrous oxide.....	0.56	0.77	
Manganese oxides.....			5.86
Sulphur.....	1.56	1.53	0.04
Sulphur trioxide.....			0.25

The slag from the basic open-hearth furnaces under certain conditions is suitable for soil conditioning. An analysis of this type of slag produced in the Birmingham district is: 8 per cent total phosphoric acid, 18.5 per cent iron oxide, 8.00 per cent silica, 5.5 per cent magnesia, 45.85 per cent lime, and 3 per cent manganese oxides⁸. No fluorspar is necessary to promote fluidity in this case.

The apparent specific gravity of the blast-furnace slag varies from approximately 2.7 to 4.2 with an average of 2.9 for granulated slag for cement manufacture in the Birmingham district. The specific gravity (apparent) of granulated slag under the process utilized in the making of Superock aggregate is 1.12 for coarse aggregate and 1.46 for fine aggregate⁸. The apparent specific gravity of air-cooled blast-furnace slag ranges from 1.80 to 2.80, averaging about 2.25. The weight per cubic foot varies from 65 to 100 lb. for dry compacted values⁹. An average value of 75 lb. per cubic foot is obtained in aggregate sizes for air-cooled slag. The weight of granulated (water-cooled) slag is 40 to 60 pounds.

Voids in slag for concrete aggregate have been determined to average 42 per cent and absorption tests average 4 per cent by weight⁹. The

objectionable impurities of slag depend upon the use to which it is to be placed. For the manufacture of slag cement, the requirements are more exacting than for some other uses. These will be discussed later. For concrete, aggregate slag of a glassy texture is rejected. Iron inclusions and tramp iron are removed before preparation.

METHODS OF OBTAINING AND PREPARATION OF SLAG

The various companies utilizing slag other than those who also produce it obtain their supplies by contracts. Sometimes the slag company contracts for the entire output of a furnace and removes it from the cooling pits with its own power shovels. For the manufacture of slag cement, the water-granulated slag is obtained by contract and delivered to the cement plant by railroad cars. Some of the iron and steel companies, either directly or through subsidiary companies, utilize the whole or part of their slag production.

The principal uses of slag call for its preparation at the furnace either as granulated or air-cooled slag. In the former, the slag as it flows from the cinder notch in the furnace falls into a trough, passing through a water spray which granulates the slag. This trough must be deep enough to prevent the slag from solidifying on the bottom and the water swift enough to sluice it along. This method, with minor modifications, is used in the Birmingham district. According to Eckel¹⁸, the success of the granulation depends upon the rapidity with which the quenching takes place after the slag leaves the furnace. The granulation process breaks the slag into small pieces and increases the porosity. There are also two chemical properties produced by this sudden chilling: (1) if the slag is of the proper composition, it becomes hydraulic and (2) the slag is partly desulphurized. The manufacture of slag cement requires careful control of the granulation steps.

A more recent development in granulation for the production of light-weight aggregate was introduced into this country from Germany and is used in the Birmingham district at the blast furnaces of the Republic Steel Co. for the manufacture of Superock. The hot slag is brought from the furnaces by cars and discharged into a patented machine, which contains paddles 5 ft. in diameter and driven at 900 r.p.m. Water under 80 lb. pressure is introduced into the machine by a pump. The resultant explosion of gases in the hot slag produces a very porous, or cellular, structure in the hardening slag. The paddles break up the lumps, which are later recrushed and sized¹⁰.

Slag-crushing Plants

The air-cooled slag for aggregates and crushed slag is dumped in a molten state into long shallow pits, sometimes dug into the top of the old

slag bank. These pits are large enough to allow an adequate period of cooling and to get the maximum amount of cracking before excavation takes place. The general practice is to excavate the slag from these pits by power shovels and transport the material to the crusher house by cars. All of the crushing and screening plants are similar in design, differing mainly in the type of equipment. Primary breaking is done by gyratory crushers set at about $1\frac{1}{2}$ in. Magnetic pulleys are used to remove tramp iron, which is returned to the blast-furnace plant. Sizing into as many as 14 grades is obtained by the use of vibrating screens. Secondary crushing is done at most plants. All plants have bins for storing the various sizes, from which the slag can be mixed in varying proportions to meet the specifications of slag aggregate and to weigh automatically. Inspectors employed by the consumers see that the specifications are carried out. The newer slag plants have dust-removal systems and provisions for preventing segregation in loading. The structures are generally of steel and concrete, or slag-block construction, and are fireproof^{11,12}.

The basic open-hearth slag sold for soil conditioning is crushed by dropping a 5-ton ball onto the large cakes. The steel is removed by a magnet. The slag is then fed to a set of rolls, which reduce it to $1\frac{1}{2}$ in. maximum. From this mill the material passes to a 1-in. revolving screen, then to a pebble mill where it is crushed to a fineness of 90 per cent through 100 mesh; then it passes through a $\frac{1}{16}$ -in. vibrating screen. The oversizes of the two screens are returned for recrushing¹³.

MANUFACTURE OF SLAG CEMENT

In the Birmingham district a portion of the blast-furnace slag from self-fluxing ore has been used for manufacturing slag cement, or puzzolan, for many years. The two plants in the district use similar processes. A generalized flowsheet of the Southern Cement Co. illustrates the practice. The water-granulated slag is unloaded at the yards of the cement plant and stacked according to analysis. From the stock yard the slag is conveyed to a rotary drier of the straight-draft type fired by pulverized coal. From the drier the slag is passed over a rougher stationary screen with $\frac{1}{2}$ -in. openings and conveyed to a surge tank, where it is fed into two roughing tube mills. The ground slag is then elevated and conveyed to a surge tank and to a charger box. Lime is conveyed through closed hydrators to another surge tank. The ground slag and lime are weighed automatically and mixed in the correct proportions in a mixing machine. This product is split into three parts and fed to three finishing tube mills. The finished product, 98 per cent through 200 mesh, is conveyed by a cement pumping unit into nine storage silos, of which the total capacity is 80,000 bbl. From these silos it is drawn into packing

tubes and packed into paper bags for shipment¹⁴. Several brands of slag cement and prepared mortar are made.

Ready-mix Concrete Plants

Several companies in the Birmingham district operate ready-mix concrete plants. In fact, the Sloss-Sheffield Steel and Iron Co. pioneered this branch with a plant built in 1924. The plant has been enlarged and modernized until now it has a capacity for mixing 1500 cu. yd. per day.

This ready-mix plant, which is representative of the practice in the district, utilizes the air-cooled slag from the company's blast furnaces. The slag is crushed and screened in the slag plant and then conveyed from the storage bins. Sand mixtures are also stocked in storage bins and conveyed to the ready-mix plant. Extensive experimentation has determined the proper mixtures of sand and aggregates to prevent segregation during the period of delivery. The sand and slag are stored at the ready-mix plant in three bins for three proportions of aggregates and in three bins for sand mixtures. The use of cement in large quantities requires special cement-handling equipment consisting of a power shovel, vibrating screen, cement pump and four storage tanks. Each tank has a capacity of 500 bbl. Both Portland cement and slag cement are used and mixtures of the two are frequently specified. The materials are conveyed to weighing hoppers and pass from them into the three mixers. From the mixers, which are of the tilting type, the product is delivered in open-body trucks¹⁴. Recently revolving mixers of 3-cu. yd. capacity mounted on truck chassis have been added to the equipment.

GENERAL USES OF SLAG

In addition to the uses of slag already mentioned, the finer sizes of crushed and screen air-cooled slags are used in rolled roofing and tar roofing. Quantities of air-cooled slag are used for railroad ballast. Slag bricks and slag block are manufactured in the Birmingham district from fine slag aggregate, both of the solid and hollow type. These have high fire-resisting qualities, good insulating and sound-resisting properties and excellent workability in cutting and nailing. The light weight of concrete products made from fine slag aggregate is an important advantage¹⁰.

Basic open-hearth slag as a soil conditioner has already been mentioned as a successful utilization of slag in the district. The production amounts to about 35,000 tons per year¹³.

Experiments have been made in the state on the utilization of slag for glass manufacture¹⁵. This use has been successful in the laboratory but has not been developed beyond that point.

In other districts blast-furnace slag has been used for the production of slag wool for both sound and heat insulation, and filter medium for acids and gases. With other substances a variety of products are on

the market. The subject of mineral wool has been fully described by J. R. Thoenen¹⁶.

More recent uses have been the utilization of slag dust as a base on unimproved city streets for future oil mix or asphalt surfacing¹⁵.

STORAGE, STOCKS AND MARKETING

Most of the producers of slag aggregate maintain substantial stocks, as their contracts call for the removal of the slag from the pits within specified time periods. This necessitates operating their crushing plants and storing their products. The cement companies also maintain storage silos from which they draw as the trade demands. Similar conditions exist with regard to other producers.

The market for slag as aggregate is somewhat limited, depending upon the availability of sand and gravel deposits within the area. Also the specifications of the various states differ in their requirements and therefore make the screening process at the sizing plant more complex, which in turn increases the cost of production. The normal producing capacity of crushed-slag plants in the Birmingham district is approximately three million tons per year. The market for slag aggregate at Birmingham is normally confined to the southeastern states, except for some fine sizes, which are shipped greater distances. The prices are listed in *Rock Products*¹⁷ in the following sizes: roofing, $\frac{1}{4}$ in., $\frac{1}{2}$ in., $\frac{3}{4}$ in., $1\frac{1}{2}$ in., $2\frac{1}{2}$ in., 3 in. These prices are given per ton f.o.b. from the producing plant or from the nearest shipping point.

Puzzolan, or slag cement is shipped all over the country, particularly some of the prepared mortar products. The crushed basic open-hearth slag for soil conditioning has a wide market and is used extensively in the agricultural states. The annual production is approximately 35,000 tons and is sold under the supervision of the agricultural departments of the different states with a guaranteed minimum of 8 per cent phosphoric acid.

PRODUCTION

The normal production of blast-furnace slag in Alabama would be approximately 2,000,000 tons annually but there are many old slag dumps and some of them are being utilized. The normal production of slag other than blast-furnace is not available. The output of slag marketed is approximately equal to the annual production.

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DISCUSSION

S. B. PATTERSON,* Lebanon, Pa.—Are high-magnesian slags used in the manufacture of Superock?

J. R. CUDWORTH.—No, the magnesia content must be less than 5 per cent.

MEMBER.—What is Superock?

J. R. CUDWORTH.—It is a light-weight, cellular concrete.

S. B. PATTERSON.—In making cement from a mixture of lime and slag, is the cement burned after the lime is added?

J. R. CUDWORTH.—No, they are intimately mixed but not calcined.

C. A. BASORE,† Auburn, Ala. (written discussion).—From the standpoint of the glass technologist, blast-furnace slag is not a promising material for the production of glass, because of the variable composition of the slag and the large amount of impurities in the slag. The most objectionable of these is iron, which is perhaps 50 times more abundant than in the raw materials ordinarily used for glass. However, we have found a way of successfully blending the slag with other glass-forming materials and thereby forming a glass that in many respects appears to be decidedly superior to ordinary soda lime glass. For instance, the normal variation in the slag from day to day apparently had little effect on the color or other physical properties of the glass, while the impurities not only were not objectionable from the standpoint of a building and structural glass but in some cases apparently were actually beneficial. If the operations are carried out in such a way that the sulphur remains in the slag during the melting the glass is black and opaque, while if the sulphur is removed during the melting (i.e., add As_2O_3) a green and transparent glass results. The high iron

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content or some other impurity apparently is responsible for the high tensile strength, unusual toughness and resistance to breakage.

The glass was made on a laboratory scale in a fire-clay crucible heated in a gas-fired furnace and later on a semiplant scale in a glass plant. Four types of glass were produced: (1) a green transparent glass, (2) a black opaque glass, (3) an opal glass, and (4) a transparent glass of relatively low coefficient of expansion. Coming to the superior quality, the slag glass has an excellent resistance to corrosive agents such as acids, hot alkali, etc. For instance, the loss in 2 N caustic soda at 100° C. for 3 hr. was only 6.5 mg. per square decimeter. Hodkins and Cousin (Textbook of Glass Technology) report two grades of chemical glass losing from 283 to 364 mg. under the same conditions. A second interesting property is the high tensile strength. As determined on threads of the glass, the average of a large number of samples was above 15,000 lb. per sq. in. Samples of soda lime glass determined in the same way gave a much lower result while Hodkins and Cousin report the average tensile strength as 2000 to 8000 lb. The toughness and resistance to breakage was a third property of much importance. Having noted the unusual resistance to breakage in all of this work, we made an effort to determine this property quantitatively. A steel ball weighing 5.45 grams was allowed to fall vertically upon a 7-in. square of the glass and the results from various heights were recorded, as follows:

Glass	Fall of Ball, Ft.	Percentage of Breakage
Double-strength window, $\frac{1}{8}$ in. thick.....	2	73
Plate glass, $\frac{3}{16}$ in. thick.....	2	60
Transparent slag glass, $\frac{3}{16}$ in.....	2	none
Black slag glass, $\frac{3}{16}$ in.....	2	none

A fourth point of much interest, particularly in connection with black glass, was the high gloss and sheen. Unlike most cheap blacks the color is a true black.

According to the U. S. Census of Manufacturers in 1929, there were 263 plants engaged in making glass. Since only 11 were reported as being in the South, apparently opportunity for the development of this industry exists near Birmingham, because of the availability of slag and building sand, the most important raw materials, the proximity to Southern markets, and cheap water transportation. (Further information can be found in *Bulletin 3* and *Bulletin 6* of the Engineering Experiment Station of the Alabama Polytechnic Institute and from articles appearing in the *Chemical and Metallurgical Engineering*, the *Manufacturers Record*, and bulletins of the Alabama Academy of Science.)

E. E. BROSIUS, Pittsburgh, Pa.—Blast-furnace slag has always interested many people as to its possible use commercially. In Germany it is a well established fact that the steel plant that does not process and sell its slag is losing a source of considerable profit. There are many different processes and means for its use in Germany, the principal one being the granulation of the slag by water followed by mixing with approximately 8 per cent of lime, compressing it into bricks, treating them in a steam retort, and allowing them to harden in the air. These bricks are rough but strong and are used throughout Germany for the construction of buildings.

At one of the large steel plants in southern Germany a granulating mill was developed into which the molten slag and water are introduced in the proper proportions to make a slag of water content of about 7 per cent, this slag being used for making the same slag-lime brick. We secured the American rights for the building of this mill and have three of these plants working, but as there is no use here for the

slag as they make it in Germany, except for Portland cement, where a dry slag is desired, we found, in experimenting with it, that we could make a popcorn slag by reducing the water admitted to the mill still further. The resultant product is a light popcorn slag of high structural strength, for which there seems to be a large field in this country, the slag being used as a light aggregate for the making of concrete shapes, and floor fill for structural buildings, as well as for column protection, roofing slabs, and many other uses. It makes an excellent soundproofing material and, as the building industry revives in the country, we anticipate a large field for it.

Another use has been found in the central West for the popcorn slag made by this process. Two load-bearing building units, 8 by 8 by 16, are made on a vibrating table, with a 2-in. slab of building stone between, the units having molds for the usual voids and the concrete being poured into these forms, making a solid 16 by 16 by 8 block while being cured. After curing, the large block is placed in a breaking machine, which splits the stone in the center, leaving two 8 by 8 by 16 concrete building blocks, with one inch of a beautiful natural stone veneer on the face. The insulating qualities of the backing make the stone more desirable than a solid stone, and since this unit makes the complete wall, the cost of an artistic stone wall is cut practically in half. The concrete made of this type of expanded slag adheres perfectly, and stands all the tests of freezing and thawing. The result may be said to be that the concrete block is at last in the high-hat class.

The popcorn slag made by this process is known in the Birmingham district as Superock, and in the Chicago District as Waylite. The product is identical. In both places it is getting a foothold as a structural concrete, when made with greater density, and is receiving recognition by the U. S. Building authorities for pre-cost joists, beams, etc.

We also recommend this granulation mill for the granulation of open-hearth slag to make a uniform product that is ideal as a burden for the blast furnace for the purpose of reclaiming the iron and manganese contained therein. This granulated slag can be passed over a magnetic pulley and the free iron and steel extracted from it, and the remainder can be used as fertilizer or for other purposes. The granulation of the slag at the furnace does away with a very large percentage of the work now necessary in taking the present cinder thimbles to the skull-cracker dump and there breaking up the slag and reclaiming what steel is available.

It is our belief that many uses will be found for the various types of blast-furnace and open-hearth slag, and a real value created for what has in the past been considered of very little value. The present major use in this country is to process it through crushing plants, sizing it as an aggregate for the building of concrete roads and concrete work in general and water granulation for use in the manufacture of Portland cement. Much money has been spent in experimentation in this country, trying to utilize this waste product commercially and we believe that eventually many uses will be found for it.

Log Washers in the Aggregate and Flux-stone Industries

By A. R. AMOS, JR.* AND S. B. PATTERSON,† MEMBER A.I.M.E.

(New York Meeting, February, 1936)

LOG washers have been used for many years in the washing of clay iron ores, phosphate rock and manganese ores, but not until the past 15 years have they been employed to any extent in the preparation of aggregate materials and fluxing stone. For a machine of such importance where tenacious clay or soft rock must be eliminated, there are few data generally available. The purpose of this paper is to set forth the results obtained from several operations, together with a few suggestions for planning a specific installation.

Where the material to be washed out consists of loose surface soil, silt, or small amounts of quickly soluble clay, there is little competition with other forms of washers, because greater power and maintenance are required for the operation of log washers, but where the foreign matter is a stiff clay, such as occurs with high-calcium limestones and in marginal gravel deposits, the ability of other kinds of washers to eliminate completely the hard clay balls is at least uncertain. Then the log washer comes into its own field, for with the breaking and churning action of the logs it will clean out the toughest clay. Even with material from the hydraulic dredge, if it contains tough clay, the log washer is desirable unless the gravel is subjected to a costly soaking and rehandling process.

Recently there has been a considerable development in other types of machines to accomplish these results but it is not within the scope of this paper to make comparisons between competitive types, but to present the best obtainable data on log washers only.

From the primitive design that gave it its name, a long log hewn from a tree, with paddles forged to shape from strap iron and attached by lag screws, the log washer has gradually been developed to the present all-steel type. As now manufactured, the log washer consists essentially of two built-up logs on which are mounted a series of paddles, carried by shafts fitted at the ends of the logs on bearings fastened outside the enclosing steel box, or trough, which is set at an angle of about 5° to 10° and into which the material is fed. Stuffing boxes at the places where the shafts pass through the box at the lower end are equipped with water-

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pressure seals to prevent grit from working out toward the bearings. (Fig. 1.)

The feed enters the trough or box near the rear, or low, end and is moved up the incline and discharged out the front or upper end by the propeller action of the paddles, which form a spiral path or screw thread around the shafts revolving in opposite direction of rotation. The lifting, churning action of the paddle blades cuts and reduces the clay balls, scours off coatings on the rock particles, and frees these impurities so that they float off in suspension with the counterflow of water over the

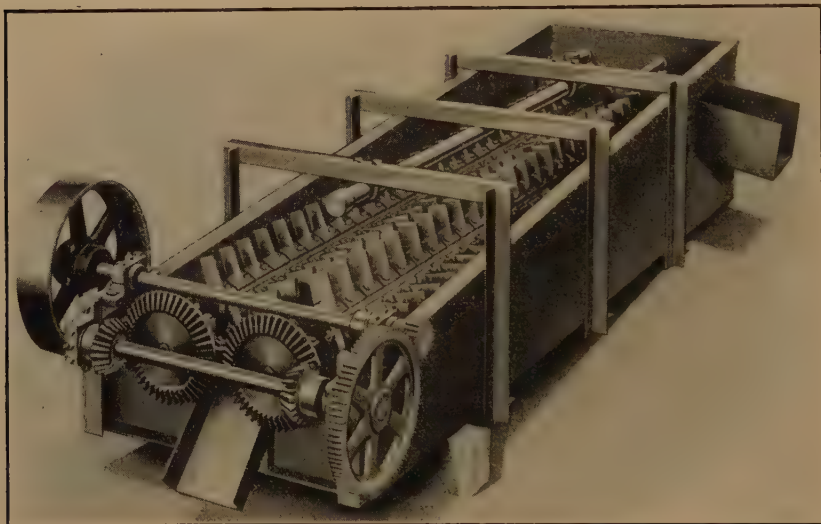


FIG. 1.—A MODERN LOG WASHER.

overflow at the lower end of the box. The logs operate at a low speed and are usually fitted with reduction gearing and belt drive from motor.

Following are the characteristics and performances of several installations operating under different conditions:

INSTALLATION No. 1

Here three log washers treat all the fines passing a $2\frac{1}{4}$ -in. square mesh that are produced in the quarrying and crushing of high-calcium limestone. The product is used in the raw mix for cement manufacture and for concrete aggregate and road stone. The washers are operated at full capacity, the amount of feed being increased (when conditions are favorable) until the motors are fully loaded, as shown by ammeters, and decreased if any particles of clay tend to carry over into the finished product.

The characteristics and performance of each log are:

Logs: double, 25 ft. long, of extra heavy steel angles.

Paddles: four rows of chilled-iron paddles, spaced 11 in. in each row, blades at 22.5° pitch, and the outside describing a circle of 3-ft. diameter.

Box: of wood. There is no wear on box because the stone makes its own bed.

Drive: 60-hp. slip-ring motor.

Production: 20.5 net tons per hour of finished product.

Water used: 67 gal. per minute.

Slimes produced: 20 per cent by weight of natural clay and fine stone.

Loss in clay and fine stone: 17 per cent.

Average power used: 59 hp. or 44 kw., equaling..... 2.15 kw-hr. per ton.

Power used for conveyor, feeder, elevator, and slurry pump.... 0.55 kw-hr. per ton.

Total power used per ton of product..... 2.70 kilowatt-hours.

Capacity can be increased, with a corresponding decrease of power per ton, by stockpiling the fines, allowing the clay to weather, but the cost of stockpiling and reclaiming the mucky and sticky material is more than any saving in power thus effected. The production per hour stated above is for average conditions where there exist a considerable number of hard lumps of clay. When washing material that is merely coated, 35 tons per hour of a clean product has been produced per washer.

Clean sand is not recovered from the slimes, for there are small grains of clay contained therein that cannot be economically separated. After remaining in the slime pond long enough to disintegrate the clay, the sand could be reclaimed by treatment in a classifier.

The costs of operation for a unit of three washers have been:

	PER HOUR	PER TON FINISHED PRODUCT
Labor. One man.....	\$0.50	\$0.008
Repair labor.....	0.08	0.001
Washer parts.....	0.23	0.004
Other parts.....	0.06	0.001
Supplies and expense.....	0.10	0.002
Power at 1.1¢ per kw-hr.....	1.83	0.030
	<u>2.80</u>	<u>0.046</u>

INSTALLATION No. 2

Conditions here are similar to those in installation No. 1, and the log washers are the same except that the feed enters the washer at a point that makes the effective length of the logs but 20 ft. The stone has the minus 1½-in. screened out before it is fed to the log washer. The foreign material is a red clay, both coating the stone and in the form of balls.

INSTALLATION No. 3

Here a double steel log washer is used to treat accumulated quarry refuse—material cast back by hand loaders when loading stone cars at

the quarry face. This refuse includes minus $1\frac{1}{4}$ -in. stone mixed with loam or clay, with clay adhering to the pieces of stone or by itself in the form of clay balls, generally ranging in size of diameter from 1 in. to 5 or 6 in. It also includes some stone larger than $1\frac{1}{4}$ -in. that is smeared with clay.

The loss in washing varies greatly, running as much as 60 per cent, with an average of 15 to 30 per cent. The amount of water used is 500 gal. per min., and the slimes are pumped from a sump to a large settling basin. When the basin becomes filled it is allowed to dry and higher retaining walls are then constructed by shovel casting and the basin is again put into service.

The characteristics of this log washer are similar to those of installation No. 1, but it is of later design. The greater production is obtained because stone of smaller size is treated, and because the clay has weathered before treatment, whereas in installation No. 1 the stone is washed as it comes from the quarry. Repair and maintenance cost is reported as approximately 10¢ per hour for the first 1100 hr. of operation.

INSTALLATION No. 4

This operation was started near the southern edge of the Morristown-Tullytown (Pa.) sand and gravel deposit, where increasing amounts of tough clay and cemented strata are encountered. The pit was opened and operations commenced with shovels, 4-yd., 36-in. gage cars, and gasoline locomotives to feed the plant hopper.

Even with a preliminary dry rotary scalper and a modern rotary scrubber on the washing screen, and using 2000 gal. per min. of wash water, there was still clay in the finished product, therefore a preliminary washing plant was installed next to the scalping plant, consisting of a pair of double log washers, the logs being 25 ft. long with cast steel paddles approximately 6 by 10 in. and describing a circle of 36-in. dia. The logs were sloped at 1 in. per foot and in wooden boxes. The two units were driven by one 100-hp. slip-ring motor through V-belts and gearing giving 25 r.p.m. to the logs. A supply of 200 gal. per min. (or about 370 gal. per ton of finished gravel) of clean wash water was supplied to each set of logs, introduced through spray nozzles distributed over a distance of 18 ft. from the intake end. In this particular case, sprays were not used near the discharge end because of feeding to an inclined belt.

The original installation permitted all material consisting of gravel, sand, clay and top soil passing 2-in. round holes in the scalping screen to go direct to the log washers, but after operation for a short period of time a change was made providing for raising the feed with a belt elevator to a 3 by 8-ft. single-deck vibrating screen equipped with sprays. The material passing the $\frac{1}{4}$ -in. square openings in this screen was flumed direct to two 60-in. by 32-ft. sand drags, which delivered the dewatered

sand to the 30-in. conveyor running to the main plant. The oversize from this screen was chuted to the two log washers. The feed to the logs averaged 60 to 70 tons per hour, of which 10 to 20 tons went over the overflow and was flumed to a settling pond. The product of the log washers was delivered by the 30-in. conveyor to the main plant for final washing and sizing. Prior discharge of sand from drags to belt provided a cushion for gravel discharge from the logs. By-passing sand around logs was practicable here as particles of clay not removed in preliminary plant were liberated and floated off in the final drag-classifier stage in the main plant.

The feed to the logs contained a considerable percentage of crushed gravel and the wear on the original cast steel paddles was fairly rapid. Replacements were made with manganese steel, which gave two to three times the service. Pinion and gear replacements were about the only other important maintenance items. The wear on the gearing was caused largely by the sand adhering to the grease. By using no grease on the gear-tooth faces, but keeping them clean with water sprays, the wear undoubtedly would be greatly reduced. Segregated costs for the log washers were not kept by this company.

This installation resulted in a very clean final product, even though the deposit was not stripped and the feed at times contained as much as 25 to 30 per cent of clay and top soil. While the material received a second washing in the sizing plant, the logs removed all of the clay larger than $\frac{1}{4}$ -in., and were the only machines tried that positively insured this result. The water supply of 200 gal. per min. for each set of logs is believed to be about the minimum and was chosen only because of the limitations of the water supply. Approximately 500 gal. per min. for each log washer was furnished in the original layout, when sand was also handled and when overflow from the logs was flumed to drags for sand recovery.

Later a 12-in. suction dredge was installed. The preliminary churning of the material in the dredge and pipe line materially reduced the work for the log washers.

INSTALLATION No. 5

This installation was dictated by conditions similar to those for installation No. 4 with a deposit of even larger clay content. The material was dug by a large dragline and transported to the plant hopper by dump cars. The installation consisted of two double log washers, which were fed all the minus 3-in. material from a rotary scalping screen. An important difference with respect to No. 4 was the use of several men as "pickers" along the belt conveyor feeding the logs to remove the larger of the clay lumps by hand. It was estimated that the clay content

in the raw feed ran 15 to 20 per cent at times, and the logs reduced this to less than the 5 per cent allowed in specifications.

INSTALLATION No. 6

This installation was made to break up and scrub gravel produced from a cemented deposit. It consists of one double log washer of the steel box type with 25-ft. logs and paddles set at 25° pitch describing a circle of 36-in. dia. The logs turn at 25 r.p.m. While usually a 60-hp. motor is recommended by the washer manufacturer, a 40-hp. motor is used by this operator. A water supply of 800 gal. per min. was provided. The slimes are disposed of by means of a sand pump. Gravel is fed at the rate of approximately 80 tons per hour. The loss in washing is 10 to 12 per cent, giving an average product of about 70 tons per hour of clean gravel. Repairs are estimated at 1¢ to 2¢ per ton of finished material.

INSTALLATION No. 7

This installation consists of a washer of the same model used at installation No. 6, but with a 60-hp. motor and the logs turning at 16 r.p.m. The feed is minus 3-in. crushed gravel with the sand removed. The clay exists as a coating adhering to the gravel pieces and, although a high-speed revolving scrubber might have proved satisfactory, the log washer was chosen, to remove any uncertainty.

INSTALLATION No. 8

This installation consisted of two double 20-ft. log washers, the logs turning at 19.5 r.p.m. and driven by 50-hp. motors. The feed was minus 1½-in. gravel with the sand removed by vibrating screens above the washers. The production was 40 to 50 tons per hour each. Here again log washers were used to insure positive results.

TABLE 1.—*Performance of Log Washers*
Figures are on the basis of one double-log washer

Installation No.	Length of Logs, Ft.	Material Treated	Size of Feed, Equivalent to Square Holes in Inches	Percentage Washed Out	Water Used, Gal. per Min.	Speed of Logs, R.p.m.	Drive, Hp.	Average Horse-power Used	Product, Net Tons per Hour
1	25	Limestone	0 -2¼	17	67	21	60	59	20.5
2	25	Limestone	1½-2½		400		100	91	25
3	25	Limestone	0 -1¼	22½	500	20	80	64	45
4	25	Gravel	¼-1¾	23	200	25	50		32.5
5	25	Sand and gravel	0 -2⅝	17½					
6	25	Gravel		11	800	25	40		70
7	25	Gravel	¼-2⅝	10	400	16	60		
8	20	Gravel	¼-1½			19½	50		45

GENERAL CONSIDERATIONS

Table 1 gives figures on the performance of log washers in the eight installations described. When installing log washers to treat material that contains varying amounts of foreign substances, it is well to keep the feed point high enough to allow the feed to enter the washers further forward than normal. Thus, when treating material that requires less action in the washers, this will reduce the effective length and save power. There is a tendency in log washers for large pieces to work back of the point of feed and jam at the back end of the box. A feed chute having for its bottom a set of grizzly bars, thus feeding the coarser pieces ahead of the finer, will correct this tendency. In installation No. 1, feed up to 3 in. could be treated without the use of grizzly chutes, but 4-in. stone worked back and this action was corrected by the grizzly chute. Log washers have been designed recently with larger paddles and greater paddle spacing, to treat this size of stone or larger.

Disposal of the slimes requires a large and convenient settling basin. Gravity flow to this settling basin would be ideal, but slurry pumps operate with a minimum of attention and repair. The power for pumping is a considerable item, for the velocity of flow must be kept high enough to prevent settling in the pipe line, and this high velocity requires more power than pumping clear water where lower velocities are the rule.

The quantity of water to be furnished raises several questions in addition to those of available supply and size of settling basin. Where the foreign material is a friable soil, a greater quantity of water will help to carry this off by its scouring action as the paddles lift new surfaces to this treatment. But where the material is entirely stiff clay and the churning action of the paddles is necessary to emulsify it, excess water does not help. In installation No. 1, pumping the slimes was necessary and any increase in water would materially increase the power required for the slurry pump.

Then there is the consideration of the loss of the finer particles carried into the overflow when using more water than necessary to produce a clean product, when these finer particles are marketable. The method of application of the water may be designed to clean the final product effectively. The sprays should be carried close to the discharge end when water in the discharge is no detriment, to give a final rinsing. This is particularly necessary when as little water as possible is to be used. The dirty water in the box is liable to leave a coating on the final product, which unless a final rinsing is used must otherwise be removed by sprays in any further screening operation. Application of additional make-up water, that is, water in addition to the sprays that may be necessary to provide sufficient volume to carry off the waste, is best made in the front

corners of the box, near the discharge end, thus providing a counterflow of clear water.

For gravel deposits, where both tenacious clay and soft stone of relatively high specific gravity must be removed, it may be necessary to provide a separate machine employing impact or pulverizing action to follow the logs.

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DISCUSSION

(*W. H. Coghill presiding*)

P. M. TYLER,* Washington, D. C. (written discussion).—The log washer is so simple in general principle and in technic of operation that it has had too little attention in the literature. This paper contributes definite data and should stimulate helpful discussion regarding its place in the treatment of mineral products as well as regarding its most economical adjustments.

Unfortunately, laboratory tests, at least through the medium of small-scale machines, are not particularly informative, therefore tests must be made in field operations. It happens, moreover, that log-washer installations, except perhaps at a very few large iron-ore mines, are at plants where rule-of-thumb methods are in vogue and where fluctuations in tonnage and nature of feed are so great as to preclude the accumulation of truly scientific observations. Comparative data, therefore, are likely to be qualitative instead of quantitative in character. On the other hand, these machines are flexible in operation and changes can be made quickly in such matters as: (1) amount of wash water; (2) overflow level; (3) height of discharge; (4) point of feeding; (5) manner of feeding; (6) rate of feeding; and even (7) speed of rotation of logs. The only operating adjustment that cannot be made fairly easily is alteration of the slope; log washers are heavy machines and once they are set in place, they are likely to stay there. However, raising or lowering the height of discharge produces much the same results as altering the slope, so, in effect, all necessary adjustments can be made within fairly wide limits.

My own experience with log washers has been confined to ore dressing, where I have used them both for removing clay and for abrading and removing soft sandstone from iron and manganese ores. Ordinarily, in service of this character, at least 50 per cent of the feed is washed away and more often the amount of waste is much greater than that of heavy solids. I recognize that this condition differs from that of washing gravel or crushed stone, but at the same time it may afford better visible evidence as to fundamental functions of these machines.

The authors have noted that a practical check upon economical operation is to crowd the machine to the point where clay balls appear in the discharge, or at least until the product is too dirty. When the ratio of solids is low, clay balls appear whenever there is insufficient heavy material in the machine and frequently they cannot be avoided by any reasonable reduction in the rate of feed or increase in water. Raising the discharge will help or, better yet, increasing the proportion of solids in the feed should be tried. This condition, of course, should not arise in the ordinary aggregate

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or flux-stone plant where heavy material necessarily constitutes the bulk of the feed but it brings out the fact that the primary function of a log washer is to grind or abrade. I do not consider it an efficient blunger for dispersing clay; it has too little beating action. On the other hand, it should not be used merely as a conveyor for turning over the material under a spray. For either of these functions, other machines are doubtless more efficient.

The functions of a log washer are to scour off sticky material and to mash down and cut up clay balls through the tumbling and grinding action of the solid material as it is lifted by the paddles. I submit, therefore, that the feed to a log washer should be properly prepared and that the washer should not be employed as a finishing machine. In other words, it should be used simply as an intermediate step in the process, except at small plants where extreme simplicity of design and minimum installation costs are more important than operating economy.

The authors have mentioned the advantage of a hydraulically mined product. Where this is impracticable, I would suggest consideration of a preliminary rotary scrubber or, if this too is impracticable, a sluice in which the material gets a preliminary wetting and tumbling over. On especially clayey ores, I have used a sloping bin lined with concrete and a nozzle similar to a hydraulic monitor. By thus pulping the ore in advance, a large reduction in clay balls can be effected at low cost, and the capacity of the log washer can be increased to a surprising degree.

Another practical trick that has not been stressed sufficiently in the literature is to allow clayey material to dry out or weather for a few days before sending it to the washer. The increased rate of dispersion of dry clay frequently offsets many times the cost of mechanically handling it a second time. Almost always it will be found to be more economical to avoid the formation of clay balls than to break them up after they are formed.

Instead of insisting that the log washer shall make a finished product, consideration should be given to inserting a short scrubber section at the feed end of a rotary screen, if this is the means for classifying the material; a regular wash trommel may be employed if sizing is to be done on flat screens or if there is a large amount of washing to be done. Where clean water is not abundant, I find there is no need for employing it except perhaps on the top spray. The log washer can be run with a thick mud overflow; apparently the only limit to the percentage of solids that can be carried in the overflow is established by the size of particle that must be retained in the solid discharge or "paddle product." If a light scrubbing follows the log washer, there is no objection to a film of mud on the discharge from the log washer.

No comparative power data are available but I feel confident that rotary mixing and scrubbing devices, ahead of log washer, and rotary washing arrangements for final cleaning are far more economical of power and that consequently the main work of the log washer should be limited to heavy scouring and breaking up tough clay balls.

W. McLANAHAN,* Hollidaysburg, Pa. (written discussion).—According to the records, McLanahan & Stone Corporation furnished the first steel log washer used for removing tough clays from limestone in 1914. This machine is still in operation. We believe that there is considerable competition in the use of logs with the use of other types of washers, even in cleaning a material that contains merely "loose surface soil, silt or small amounts of quickly soluble clay." The reason for this statement is that it is believed that the steel logs are in most, if not all, cases as economical and efficient as any other type of washer, even when washing the materials above mentioned. With almost any other washer there is the added expense of pumping more water to produce a similar result. Many of the other types of washers break the rock and round the

* Director of Sales, McLanahan & Stone Corporation.

corners, which is objectionable. The capacity of the modern log washers has been so greatly increased and the power requirements reduced so much that they can now be used economically for any type of washing.

In describing log washers, it should be added that they are often built with one to four logs in the same box.

McLanahan & Stone Corporation cannot recommend manganese steel for the paddles as economical when compared with the standard Macmetal paddles.

"For gravel deposits, where both tenacious clay and soft stone of relatively high specific gravity must be removed" it would seem advisable to remove it before washing instead of afterward, so that the material may be washed after crushing. Crushing before washing usually is done by single-roll crushers, because they will crush mud as well as rock. As a rule, objectionable soft rock can be eliminated by the proper style and arrangement of paddles in log washers. Actual tests made by state engineers in two western states show that objectionable soft rock can be reduced from a content of 11 per cent to less than 1 per cent by the use of steel log washers. This had not been done before by the use of any other machine without degradation of the material.

This company has been building these machines for over 50 years for washing iron, manganese, zinc and gold ores, diamonds, phosphate, fluorspar, peat, barite, ocher, limestone, sand and gravel, etc. It took out the first of its many patents on steel logs about 1890. Patterns are still available for building the light or extra heavy single or double wood or steel log washers, in many different styles, in any length up to 35 ft., though as a rule the old-style, single driven, open-gear logs have been displaced by the all-welded, square, double driven logs, with all gears held in one closed housing and running in oil. The new log and paddle arrangement, with overflow and discharge spouts, and the logs themselves easily and quickly adjustable to meet changing conditions, have revolutionized log capacity, power requirements, cleaning efficiency and costs.

There is no rule-of-thumb method for handling different materials in log washers. It must be decided whether the work can be done in a single or a double log, or whether more are necessary, and whether they should be set in tandem or in series.

The amount of waste may vary from 2 or 3 per cent to 90 or 95 per cent, which largely determines the amount of water required, the length and arrangement of logs, and the speed at which they should be operated.

The angle, width, length, location and number of paddles, and slope at which the logs are set must be determined for each particular installation.

Horsepower required depends upon the specific gravity, size and shape of materials to be recovered and waste to be removed.

Total cost, including interest and depreciation, for washing different materials may therefore vary anywhere from 2¢ per ton up to possibly as much as 6¢ or even 8¢ per ton of material recovered, depending upon plant conditions and the varying factors mentioned above.

G. F. METZ,* York, Pa.—In an operation in which we use a conical scrubber, we have adopted the cone principle of the Hardinge mill to the scrubbing problem. We have followed log washers at the Southern Phosphate Mines Corporation, to free the phosphate-rock pebbles from tough adhering clay. Our scrubber cleans the log-washer product. We operate our scrubber with a thick pulp. With the great mass of material in the conical scrubber, it can be operated under those conditions. We have to rinse the material as it leaves the scrubber, and I should think the same would be true of the log washer. Material is run straight through our machine and, of course, the action due to the lifters inside the scrubber is entirely different from that in a log

* District Manager, Hardinge Company, Inc.

washer. The material in our scrubber is given a churning action in a bath of water. Fifty to sixty gallons of water per minute per ton per hour is about our average water ratio.

A. R. AMOS.—There is just one point that I want to mention regarding the thickness of the overflow; I believe the log washer is fundamentally an intermediate step in sand and gravel, particularly, and while I have not seen it tried I believe the water coming from the final rinsing screen stages, which generally take place, could be carried back and used in the log washer. I approve of carrying a pretty thick mixture, because usually a rinsing afterwards is required to produce a finished gravel product.

Preparation of High-specification Sand at the Grand Coulee Dam

BY ANTHONY ANABLE,* MEMBER A.I.M.E.

(New York Meeting, February, 1936)

THE definite trend to stricter specifications with respect to hydraulic concrete has become increasingly manifest in the last six years or so; but it remained for the vast reclamation projects of the Bureau of Reclamation of the Department of the Interior of the last few years to focus attention on this trend, and to demonstrate how thousands of tons of concrete-making aggregates can be produced continuously and automatically with a control of fineness comparable with that obtained with small samples in well equipped private and governmental laboratories.

Pioneer work in this direction was done, to be sure, by private companies, notably the Kaiser Paving Co. of San Francisco, prior to the present vast public works program. Then in 1931 came the first great aggregates-preparation plant of the Six Companies, Inc. to supply construction materials for Boulder Dam¹ in Nevada, and now, in 1936, the aggregates plant of the Mason-Walsh-Atkinson-Kier Co., general contractors on the Grand Coulee dam in Washington.

At Boulder dam and to an even greater degree at Grand Coulee dam, sand production is recognized as probably the most important single element affecting the physical characteristics of the concrete. For in the construction of dams accurate grading of the fine aggregate has considerable bearing on the attainment of a uniform and homogeneous concrete of a high degree of workability and placeability and, at the same time, combining high density and low cement content. Low cement content is of value not only from the standpoint of economy but also because the use of an unnecessarily rich concrete causes certain undesirable characteristics to develop.

This paper will relate how at Grand Coulee dam methods that are more advanced than ever before are producing four sizes of gravel and a sand having a fineness modulus of 2.5-3.0. The design capacity is 1000 tons an hour. Significantly, the methods and equipment employed are broadly those that have been used for a generation in metallurgical mills for accurately sizing and grading both precious

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¹ Aggregate Production at Hoover Dam. *Trans. A.I.M.E.* (1934) **109**, 397-417.

and base-metal ores. Reference to the vast Columbia River Basin reclamation project, of which the Grand Coulee dam and its sand-preparation plant are parts, will be omitted because this subject has been adequately treated in the current technical press².

THE PIT

Brett Banks, slightly over a mile from the site of the dam, is the source of all the materials washed and graded in the aggregates plant. Gravel deposits of the desired character lie both above and below a layer of sand. Overburden is stripped to a depth of a yard or so and excavation is by means of electric shovels, equipped with scoops of 5 cu. yd. capacity. Raw aggregates move directly from the shovels to storage at the preparation plant on an elaborate system of conveyor belts and without intermediate haulage by motor truck or railroad car.

THE SPECIFICATION

The Government specification calls for the segregation of the run-of-pit material into four sizes of gravel and a single blended concrete sand, as indicated in Table 1.

TABLE 1.—*Government Specification for Aggregates*

	INCHES Through 6	INCHES On 3
Cobbles.....	3	1½
Coarse gravel.....	1½	¾
Medium gravel.....	¾	¼
Fine gravel.....		
Concrete sand of fineness modulus 2.5–3.0, and having a screen analysis approximating that shown in Table 2.		

The peak in the concreting schedule is 12,000 cu. yd. a day, equivalent to about 16,000 tons of mixed aggregates.

COARSE AGGREGATES

Upon arrival at the plant, raw stock passes on to two 6-in. scalping screens, preceding a gyratory crusher, which reduces all run-of-pit stock to minus 6 in. From this point on the screening plant is built in two identical sections, independently operated to insure flexibility. All screens are sprayed, first with reclaimed water and then with fresh water.

On the third and top operating platform of the screening building there are four double-deck vibrating screens, two per section. Here cobbles, through 6 in. and on 3 in., are prepared on the upper deck, and coarse

² *Eng. News Record* (Aug. 1, 1935).
Rock Products (Jan., 1936).
Pit & Quarry (Jan., 1936).

Pacific Coast Builder (Dec. 7, 1935).
Compressed Air Magazine (Sept., 1935).

gravel, through 3 in. and on $1\frac{1}{2}$ in., on the lower deck. Undersize gravitates to the floor below, where eight double-deck vibrating screens, four per section, are installed. These eight duplex screens produce, first, a medium gravel, ranging from $\frac{3}{4}$ to $1\frac{1}{2}$ in., and second a fine gravel, between $\frac{1}{4}$ and $\frac{3}{4}$ in. Each of the four grades of gravel is carried by belt conveyor from its screen to its individual storage pile. A tunnel passes directly under these storage piles to facilitate the transfer of gravel to the concrete-mixing plant.

The undersize from the last screen and the bulk of the wash water flows to a dewatering drag. Here the sand is settled and removed by the drag flights while the greater part of the wash water overflows across a weir and passes thence, with other wash water, to the water-recovery system, which will be described later in this paper. The function of the dewatering drag is to bleed off the greater part of the wash water used on the screens, the presence of which in such excessive amounts would jeopardize close control of the delicate sand-grading operation that follows. No attempt is made to grade sand in the drag, although, to be sure, a good deal of silt and ultrafine sand is removed with the excess water.

FINENESS MODULUS AND HYPOTHETIC SCREEN ANALYSIS

A fineness modulus of between 2.5 and 3.0 was specified for the sand. Fineness modulus is the sum of the cumulative per cent weights on standard sieves 4, 8, 14, 28, 48 and 100 mesh divided by 100.

Table 2 shows a distribution of grain sizes that was found to give excellent results, and which yielded a final blended sand having the specified average fineness modulus of 2.75.

TABLE 2.—*Distribution of Grain Sizes*

Mesh	Per Cent Wt.	Cum. Per Cent Wt.
8	12	12
14	20	32
28	24	56
48	24	80
100	15	95
-100	5	—
Total.....	100	275

Fineness modulus..... $275/100 = 2.75$

To produce the required amount of finished aggregate, including all four grades of gravel and a single grade of high-specification sand, it was calculated that there would be required 278 tons per hour of blended sand. Since, in the run-of-pit raw material ultra fine sand and silt predominated, it was estimated that from 400 to 600 tons of raw sand of average com-

position would have to be processed in the plant to yield these 278 tons. Carrying the calculations one step further, it was seen that the quantities indicated in Table 3 would be needed.

TABLE 3.—*Tonnages*

	TONS PER HR.
Total raw sand to sand plant.....	400
Size tonnages prepared:	
MESH TONS PER HR.	
+ 8 33	
14 56	
28 67	
48 67	
100 41	
-100 13	
Total blended sand.....	277
	—
Total spoils to waste.....	123

SAND-PREPARATION PLANT

As described, the sand that enters the aggregates plant is finally dewatered in a drag-flight settler, and this dewatered sand is fed to the first of three bowl classifiers.

Classification Step No. 1—Coarse Sand

The first classifier is a heavy-duty, Dorr turret bowl classifier, 8 ft. wide, 35 ft. long, and equipped with a bowl 13 ft. in diameter.

The discharge from the drag dewaterer, diluted with water to give a free-flowing pulp, flows into the central, semisubmerged feed well of the bowl, which reduces the velocity of the incoming pulp stream and distributes it radially across the bowl. The bowl rakes are run at such a speed and the dilution of the pulp in the bowl is adjusted to such a point, that relatively coarse sand—4 to 28 mesh—settles out upon the bowl bottom while fine sand—through 28 mesh—remains buoyed up in the pulp and consequently overflows a peripheral weir extending around the complete circumference of the bowl.

Broadly speaking, the greater the speed of the bowl rakes and the lower the dilution of the pulp, the coarser is the mesh of separation, and vice versa. The size of the bowl is a function of the capacity to be handled or the fineness of the separation to be made; the larger the bowl, the greater the capacity or the finer the separation.

The coarse, plus 28-mesh sand, deposited on the bowl bottom, is raked to the center of the bowl by inclined plow blades attached to the radial arms of the bowl mechanism. At the center of the bowl bottom there is an adjustable, rectangular opening, connecting the pulp in the

bowl compartment with that in the reciprocating rake compartment directly below. Through this opening the coarse sand falls, but is opposed in its downward movement by a counter flow of rising water, introduced in the reciprocating rake compartment. It is the function of this counter flow of water to wash the coarse sand free of any adhering fine particles and to sweep these adhering fines back into the bowl and out of it, over the peripheral weir.

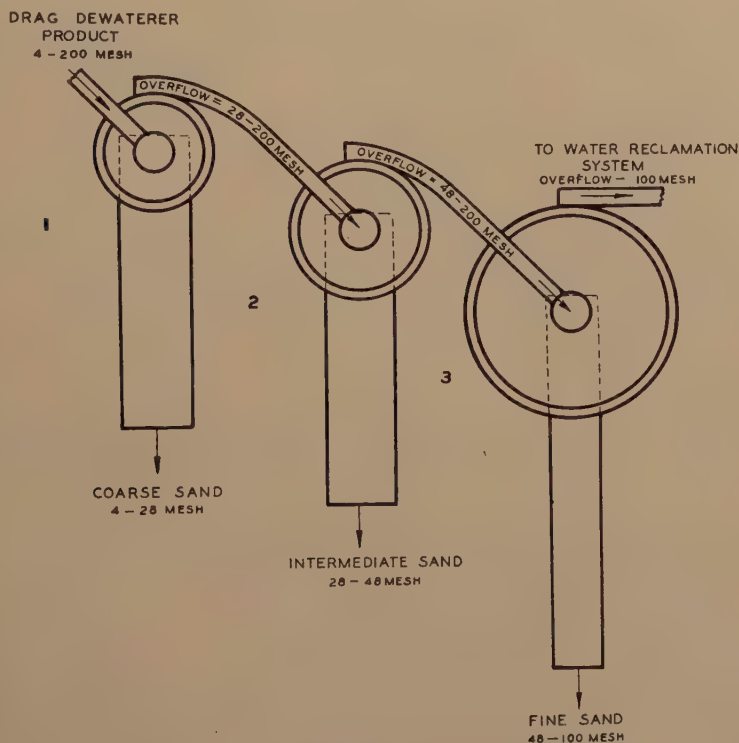


FIG. 2.—FLOW SHEET OF SAND-GRADING PLANT.

In the reciprocating rake compartment, the sand is subjected to a jiggling action by the rakes, which serves the dual function of jiggling out any still occluded fines and of advancing the coarse sand, step by step up the inclined deck to the point of discharge. Water is sprayed on the sand at its point of emergence from the pulp. On the above-water portion of the inclined deck, surplus water is drained off. The greater the rake speed, the greater the capacity and the jiggling action, and also the coarser the separation made.

Operating data are given in Table 4. These were composited in December, 1935, and are believed to be fairly representative of average operating conditions.

TABLE 4.—*Operating Data, Classification Step No. 1*

	TONS PER HOUR
Feed.....	400-500 ^a
Coarse sand discharge.....	250
Fine sand overflow.....	250 ^a

Screen Analysis Product

Mesh	Per Cent	Per Cent Cum.
4	1	1
8	22	23
14	27	50
28	27	77
48	17	94
100	5	99
-100	1	
	100	

^a It was extremely difficult to get adequate samples at these two points, therefore feed tonnage was estimated and overflow tonnage secured by differences.

Classification Step No. 2—Intermediate Sand

The second classifier is also a heavy-duty Dorr turret bowl classifier of the same length and width as the first one, but equipped with a larger bowl; 16 ft. in diameter. Feed to it is the overflow from the first classifier

TABLE 5.—*Operating Data, Classification Step No. 2*

	TONS PER HOUR
Feed.....	250 ^a
Intermediate sand discharge.....	70
Fine sand overflow.....	180 ^a

Screen Analysis Product

Mesh	Per Cent	Per Cent Cum.
14	3	3
28	15	18
48	44	62
100	38	100
-100	0	
	100	

^a See note under Table 4.

and its overflow, in turn, becomes the feed to the third and last of the three classifiers. The rake product, of course, is the intermediate grade of sand. Operating data, composited in December, 1935, are given in Table 5.

Classification Step No. 3—Fine Sand

The third and last classifier is of the same type and make as the others, but is narrower because of the lighter load to be raked, and has a larger bowl. Its dimensions are: width 6 ft., length 39 ft. 6 in. and bowl diameter 25 ft. From a feed consisting of the overflow from the second classifier, it produces, as a rake product, a very finely divided grade of sand and a waste product, as an overflow, which is retreated elsewhere for water reclamation. Operating data, composited in December, 1935, are given in Table 6.

TABLE 6.—*Operating Data, Classification Step No. 3*

	TONS PER HOUR	
Feed.....	180 ^a	
Fine sand discharge.....	90	
Ultra fines in overflow.....	90 ^a	
Screen Analysis Product		
Mesh	Per Cent	Per Cent Cum.
28	1	1
48	6	7
100	60	67
200	31	98
-200	2	
	100	

^a See note under Table 4.

Blending Sand

The rake discharges from these three classifiers are segregated in separate piles of "coarse sand," "intermediate sand" and "fine sand." As mentioned earlier in this paper, the final, blended concrete sand is made up by mixing different amounts of these three different grades of sand in such proportions as will yield a composite having a fineness modulus of between 2.5 and 3.0. Consequently this proportional relation may vary from day to day, as the character of the feed to the plant varies. Even with these day to day variations in feed tonnage and feed character, the blended sand day after day meets the specification almost exactly.

Table 7 gives calculations showing the distribution of sizes and the fineness modulus if all of the sand from all of the classifiers were mixed without any proportioning. This has been done to show how closely the blending is done automatically in the classifiers themselves and how little additional blending has to be done from the stockpiles.

TABLE 7.—*Tons Produced per Hour*

Mesh	No. 1 Classifier	No. 2 Classifier	No. 3 Classifier	Total	Per Cent	Per Cent Cum.
4	2.5	—	—	2.5	0.6	0.6
8	55.	—	—	55.	13.4	14.
14	67.5	2.1	—	69.6	17.0	31.
28	67.5	10.5	0.9	78.9	19.5	50.5
48	42.5	31.5	5.4	79.4	19.6	70.1
100	12.5	26.4	54.	92.9	22.6	92.7
200	2.5	—	27.9	30.4	7.4	—
—200	—	—	1.8	1.8	0.5	—
Total.....	250.0	70.5	90.0	410.5	100.6	258.9

Fineness modulus: $258.9/100 = 2.59$ inches.

The above is to be compared with the following, showing a typical distribution of sizes in the run-of-pit sand.

Raw Sand Grading

Mesh Distribution	Per Cent	Per Cent Cum.
4- 20	34.3	34.3
20- 48	29.1	63.4
48-100	28.2	91.6
—100	8.4	100.0
	100.0	

A review of the data presented in Tables 4 to 7 shows that the separations being made by the three classifiers are in the following ranges: (1) coarse sand, 77 per cent 4 to 28-mesh, inclusive; (2) intermediate sand 80 per cent 28 to 48-mesh inclusive; (3) fine sand, 74 per cent 48 to 100-mesh inclusive.

Sand Disposal

The three grades of sand from the three classifiers are stockpiled by automatic reversing trippers, so that the sand in each pile is automatically blended to smooth out any irregularities in the plant. Adjustable feeders discharge the sand from the piles on to belt conveyors running in tunnels, delivering the three grades of sand first to a sand mixer and then to a final stockpile. By adjusting the three feeders, the three grades of sand are apportioned in the desired ratio to yield a blended sand of the desired fineness modulus.

All aggregates, both gravel and sand, are transferred from stockpiles at the aggregates plant to the two concrete-mixing plants by belt convey-

ors. One mixing plant is on the east bank of the Columbia River and the other on the west bank. Cement is delivered pneumatically.

WATER RECOVERY AND CLARIFICATION

The amount of water used for washing purposes on the gravel screens and in the sand classifiers is great at a plant of this type, treating 1000 tons per hour and upwards of silt-laden material. At rated capacity the water requirements may easily reach 20,000 gal. per minute. Furthermore, the aggregates plant is about 650 ft. vertically above the river level, all of which would have entailed a heavy pumping cost if all of the water had to be drawn from the river and used but once for washing purposes.

Therefore an elaborate water-recovery system was decided upon, so that the bulk of the spent wash water might be clarified for re-use and pumpage reduced simply to make up for losses in the system. It now appears evident that only a small portion of the total water required will have to be drawn from the river with high-lift pumps.

The water-clarification plant is somewhat below the level of the sand classifiers, so that the overflows from No. 3 classifier and from the drag-flight dewaterer can flow to it by gravity. As explained, all of the wash water used both in the aggregates and sand plants leaves the system at these two points, laden with sedimentary silt and sand too fine for use, largely minus 100 mesh.

The plant consists of two concrete settling tanks, one roughly elliptical in shape and the other round. The long axis of the larger tank is 250 ft., the short axis 125 ft. and the depth at the side 11 ft. The smaller tank is 125 ft. in diameter and 11 ft. deep at the side.

In the larger elliptical tank two Dorr clarifiers are installed, each 125 ft. in diameter, while in the smaller round tank a single Dorr clarifier of the same size is used. These clarifier mechanisms are of a new design, perfected in 1935, in which the entire mechanism and drive unit is supported on a central stationary pier. These are also the first units ever to be provided with a new overload-relief device that operates automatically to raise the rake arms when a heavy overload occurs and to lower them again to their regular operating positions when normal operating conditions are again attained. The actuating force is the torsional load imposed which causes the rake arms to swing bodily upward, pivoting at the center of the tank.

The clarifier mechanisms revolve at a speed of 4.6 revolutions per hour. At each revolution the rakes sweep the tank bottom, plowing the settled silt and sand into an annular depression at the center, from which they are drawn off as a thick sludge to waste. Feed from the drag dewaterer and the classifier enters from a segment of the periphery of the

tank and, after depositing its load of suspended solids, overflows in a clarified state through a segment diametrically opposite.

TABLE 8.—*Feed Characteristics*

Date	Time	Solids, Per Cent	Plus 100-Mesh, Per Cent
12/10/35	9:00 a.m.	4.65	1.9
12/10/35	10:15 a.m.	9.2	2.0
12/10/35	2:30 p.m.	8.75	2.0
12/11/35	9:30 a.m.	9.15	1.4
12/11/35	12:30 p.m.	8.3	1.8
12/12/35	2:45 p.m.	4.8	1.7
12/12/35	3:10 p.m.	7.6	
12/13/35	1:30 p.m.	8.3	
12/16/35	10:00 a.m.	4.35	3.0
12/16/35	11:30 a.m.	6.3	1.5
Five-day average..		7.14	1.9

TABLE 9.—*Operating Data, Clarification Plant*

	FEED	TONS PER HOUR ^a
Water, 20,000 g.p.m.....		5000
Solids:		
Drag dewaterer overflow.....		60
No. 3 bowl classifier overflow.....		90
Solid content, 3 per cent solids.....		150

^a Approximate only. Determined by measuring cross-sectional area of pulp in trough, velocity of flow and percentage solids.

Table 8 shows how greatly the feed characteristics vary from day to day and from hour to hour. The rough average figures of 7.14 per cent solids and 1.9 per cent coarser than 100 mesh must therefore be considered with caution. More accurate over-all tonnage comparisons made later and reported in Table 9 indicate that the average solid content of the feed to the clarifiers is 3 per cent, not 7.14 per cent as shown by these grab samples. When steady 24-hr. a day operation is attained these wide variations should be ironed out and absorbed as the material flows through the system.

UNDERFLOW AND OVERFLOW CHARACTERISTICS

The clarifier underflows, consisting of settled solids and water, are removed continuously from points directly below the centers of the three mechanisms and piped to a spoils pond on low ground. No. 1 clarifier, round tank, gives 25 per cent solids; No. 2 clarifier, elliptical tank, 35 per cent; No. 3 clarifier, elliptical tank, 25 per cent; average, 28 per cent solids.

A greater percentage of solids is anticipated when the entire plant is placed in continuous operation at full rated capacity.

The overflow from the clarifiers is substantially free from suspended solids, although no attempt is made to produce a crystal clear water. It is, however, entirely satisfactory for re-use for washing purposes.

TABLE 10.—*Power Consumption*

	Installed Power, Hp.	Consumed Power, Hp.
No. 1 clarifier, round tank.....	5	1.25-2
No. 2 clarifier, elliptical tank.....	5	1.25-2
No. 3 clarifier, elliptical tank.....	3	0.75-1.25
Total.....	13	3.25-5.25

SUMMARY

The Grand Coulee dam sand-washing plant is not only the largest in the world but probably also the highest development to date in the realm of production of high-specification sand.

From a run-of-pit material, predominatingly high in fines and varying daily and even hourly in composition, a clean, sharp sand is consistently produced having a fineness modulus of between 2.5 and 3.0.

The three classified sands produced, when blended in their natural proportions, yield a sand of the desired specification.

As a safeguard, automatic proportioning devices are installed so that the ratios of the three different sands may be varied at will to alter the fineness modulus of the blend as desired. In addition to the variations provided by the automatic sand-proportioning devices, variations can be effected within the three classifiers themselves by several adjustments, such as changes in the speed of the rakes, changes in the amount of water used, et cetera.

A mechanically operated water-recovery system permits most of the wash water to be salvaged and re-used, thus greatly reducing high-lift pumpage; and removes the suspended impurities in a compact form for disposal as spoils.

CONCLUSIONS

Flexible, mechanically operated sand-preparation plants, smaller than the one at Grand Coulee dam but similar in design, should be ideally adapted to the preparation of rigid specification, which today is being more and more urgently demanded.

Given a relatively good sand pit and a system of sizing and blending both in the classifiers and in the succeeding proportioning feeders, it should be entirely feasible to produce not one grade of sand, as at Grand

Coulee, but substantially the whole range of sands specified by contractors and builders.

ADDENDUM

Since the time of presentation of this paper some additional data have been secured from the field that should properly be included to reflect the increased scale of operations as of April 1, 1936.

Although the sand and gravel preparation plant was designed to handle 1000 tons per hour maximum, normal tonnage has now been increased to about 1700 tons per hour and at one time a peak of 2500 tons per hour was maintained for a day.

Experience has shown that the run-of-pit material contained greater amounts of sand—particularly very fine sand—in proportion to gravel, than initially supposed. Consequently, to obtain the desired amount of gravel per hour, more tonnage has had to be put through the gravel plant and excess sand wasted, either at the drag dewaterers or at the classifiers, to the extent of 1000 to 1500 tons per hour. Fifty to 65 per cent of the drag discharge is sent to waste, but even so the classifiers are operating at 25 to 50 per cent, at least, of the design capacity. The three 125-ft. clarifiers have been called upon to clarify for re-use all of the extra wash water, with its greatly increased silt load, with the result that the three sludge discharges are running 400 tons per hour on a dry basis as against 107 tons per hour, design capacity.

It is believed to be significant that heavy construction, insisted upon by the general contractor for the dam, has justified itself many times over by reason of its ability to absorb overloads that never were contemplated during design.

DISCUSSION

(*W. H. Coghill presiding*)

MEMBER.—What conditions determine the amount of water circulating—what governs the amount of water?

A. ANABLE.—The 20,000 gal. referred to is the amount of circulating water. This is determined not so much by the requirements in the sand-preparation plant as by the requirements in the gravel plant. There are four stages of screening, preparing four grades of gravel. On every one of the screens, water is applied. Most of the water passes through the screens, backs up through the system, and finally reaches the sand plant. The 20,000 gal. per minute is determined by the requirements of the gravel-screening operation.

MEMBER.—What are the relative advantages of the elliptical and the circular tanks?

A. ANABLE.—The advantage is one of arrangement rather than of capacity. At the large water-purification plant that the Department of the Interior is to build in Arizona for purifying Colorado River at the head of the All American Canal, 8,000,000,000 gal. of turbid water is to be clarified per day. There are to be four large

rectangular tanks, each with a number of clarifying mechanisms installed. This Coulee Dam installation is a test plant to determine at the outset the feasibility of locating a number of clarifiers in one tank in this way. At Coulee Dam we have three clarifiers. At the All American Canal there will be 72. The elliptical and the circular tanks handle the same class of material and have been checked one against the other. The larger tank handles twice the amount of the smaller tank, but handles the same amount of feed per square foot of area.

It is true that in feeding in tandem, feeding along the long axis of the tank, the No. 1 clarifier gets the heaviest load because this material is deposited near the feed end, and the second machine will get a light load. That is showing up in the power readings. No. 1 machine is running about 40 per cent more power consumption than the second.

MEMBER.—What happens to the material that builds up in the area not covered by the raking action?

A. ANABLE.—It just builds up until it reaches the angle of repose and then slides down. It is, however, only the fine, light material and this gives the least amount of rake loading.

MEMBER.—Does the material that slides off overload the rakes?

A. ANABLE.—It has not been running long enough yet. But the construction of the machine is such that when a heavy overload is imposed the rake arms swing up automatically from the center to clear the obstruction.

T. M. PRICE,* Oakland, Calif. (written discussion).—I should like to raise the question of the effectiveness of the sand mixers, as designed for the Coulee Dam. It has been my contention that because of the extreme importance of securing a properly graded fine aggregate for concrete, the method of mixing the various sizes was likewise most important; and that this operation was most effectively done when the mixing was performed in a saturated or nearly saturated condition. Fine sand has a tendency to cling together in lumps, largely because of its small particles. This condition is helped by any clay or silt content, which usually is to be found in the fines or minus 28-mesh rather than in the coarser sands. It seems to me that this material is more easily broken up and distributed throughout the mass when it is saturated than in any other condition, not excepting a perfectly dry condition. At the Boulder plant, this mixing was performed in a large Dorr bowl classifier, which also served as a collection unit for the 100-mesh as well as a final dewatering unit. I believe the records of the Bureau of Reclamation will prove that the mixing was very effectively accomplished. It would be interesting to quarter some specimens of the Grand Coulee sand to ascertain whether the grading was equal in all parts of the mass, so as to determine the effectiveness of the mixers.

* Henry J. Kaiser Co.

A Method for Estimating the Efficiency of Pulverizers

BY RAYMOND WILSON*

(New York Meeting, February, 1937)

GRINDING costs are an important item in cement manufacture, and the cost of power is one of the large items in grinding costs. Even where power is of secondary importance, cost items dependent on mill capacity are influenced by the efficiency with which the mill uses power. Tube mills typify the grinding mills most generally used in cement plants, though other types are in use. Tube mills have been widely criticized as being extremely inefficient, and this feeling regarding them has led to many attempts to design more efficient grinding equipment. Other types may be an improvement over tube mills, but the improvement has certainly not been revolutionary, at least as applied to clinker grinding. The very fact that there is doubt regarding the relative efficacy of various types of mills is an indication that an adequate basis for an analysis of performance is lacking.

The development of methods of analyzing mill performance has been hampered by the lack of a suitable method for measuring the work actually done in pulverizing. There has been no acceptable standard to which mill performance might be compared. As a consequence, it was not known whether the margin of possible improvement in pulverizing operations was large or small. Even direct comparison of different types of mills was seldom possible because most plants had only one type of mill for each type of service. Comparisons between plants were difficult because of differences in the grinding characteristics of various clinkers.

Some recent developments complete a chain of circumstances that place the cement industry in a particularly fortunate position for the study of its clinker-pulverizing operations. Modern investigators of grinding phenomena have generally agreed that the best measure of mill performance is the production of new surface. For most materials this measure of performance is largely theoretical and has a rather remote connection with the practical aims of the grinding operation. In clinker grinding, on the other hand, surface area is intimately connected with the properties conferred on the cement by grinding. Moreover, the determination of surface area is rapidly coming into routine use as a test

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of fineness. Thus, the test which has been difficult to make and which has only theoretical value for most materials, is easy, practical, and direct when applied to clinker grinding. Clinker is almost unique in this respect, among all materials that are subjected to grinding.

While actual mill accomplishment is now readily subjected to measurement, a complete rating of a mill demands the establishment of a standard of perfect accomplishment. The ratio of actual to perfect accomplishment is the proper basis for estimating efficiency of any process. The establishment of such a standard necessitates the development of a test of the grinding properties of a clinker which reveals the amount of energy needed to produce a certain amount of new surface. This paper describes a new grindability test, and shows how it may be applied to a study of the performance of laboratory and commercial grinding mills.

IMPACT GRINDABILITY TEST

Many grindability tests have been proposed and used. The new test described here has some distinct advantages which warrant its presentation. The essentials of any grindability test are a means of pulverizing the sample under some sort of controlled conditions and a means of measuring the amount of pulverizing accomplished. Many grindability testers are simply batch mills in which energy is continuously applied to a sample of material. At best, such devices give only relative information, since even if the energy input is known, there is no way of knowing how much is applied to pulverizing the sample. They therefore require the use of some material as a standard of comparison. Another disadvantage of devices of this type is their general tendency to change in efficiency during a test. The extent of the change may vary from sample to sample, or even between duplicate tests on the same sample.

The experimental difficulties in the determination of the surface of fine powders have frequently led to the use of highly inaccurate methods of estimating surface from the sieve analysis. Difficult or unreliable surface-area measurements have been a serious obstacle to the development of a really satisfactory grindability test.

Some investigations by Gross and Zimmerley¹ resulted in the development of a method which was not subject to the objections suggested above. Their apparatus was an impact crushing device, in which the impact was supplied by a freely falling weight. This device was practically frictionless and provided an accurate measure of the energy applied to the sample. The surface of the product was measured by a rate of solution method which apparently was fairly accurate and reliable—certainly much more so than calculations based on sieve analyses.

¹ J. Gross and S. R. Zimmerley: Crushing and Grinding, III—Relation of Work Input to Surface Produced in Crushing Quartz. *Trans. A.I.M.E.* (1930) **87**, 35.

This method of surface measurement required expert manipulation and was rather slow. It is doubtful whether it could be used successfully on cement clinker. More recently, Wagner² has developed a method of surface measurement that is especially adapted to cement and is rapid and reliable. The Conservation Bureau of the Portland Cement Association (unpublished reports) has conducted researches that establish the propriety of using an impact crushing method as a grindability test, and others that provide some necessary correlations of surface measurements of fine and coarse particles.

The grindability test developed by the author is a combination of the pulverizing method of Gross and Zimmerley and the surface-measurement method of Wagner. The pulverizing apparatus and method have required some modification to fit them to the specific needs of a clinker-grindability test. The test gives the grindability of the clinker directly in terms of surface produced by a unit of energy applied under conditions about as favorable as can be imagined.

The crushing apparatus for the impact grindability test is sketched in Fig. 1. It consists of an anvil, a plunger, and a guide tube. The guide tube is a 2-in. length of 2-in. brass tubing. The anvil and plunger are hardened tool steel (Ryerson VD). The anvil is machined flat on the lower surface and to the shape shown on the upper surface. The center boss and the peripheral ring project $\frac{1}{4}$ in. above the annular depression between them. The boss is machined to a diameter that fits closely but not tightly inside the guide tube. The plunger is machined to the same diameter as the boss on the anvil, and to a length of 2 in. The entire surface of the plunger and the surface of the boss were machined smooth and polished with fine emery after hardening. The anvil is supported on a 2-ft. length of 5-in. diameter mild steel bar set upright in a concrete base. A set of three small rods of soft metal is placed between the anvil and the column. Impact is supplied to the plunger by a $2\frac{1}{2}$ -in. polished steel ball of the sort used in ball bearings. The ball release is an electromagnet provided with a steel ring which centers the ball at the same place for successive drops. The magnet is supported in a frame that can be set to allow the ball to drop various distances and is placed so that the ball strikes the center of the plunger when the anvil is set with its sides flush with the sides of the column.

In the test procedure that has been adopted as the standard, a 1-gram portion of 20 to 28-mesh crushed clinker is subjected to the impact delivered by a 24-in. fall of the ball. The clinker is spread uniformly over the surface of the anvil with the brass guide in position to serve as a retaining ring. This quantity of clinker makes a layer one particle thick, covering more than half the crushing surface of the anvil. The plunger

²L. A. Wagner: A Rapid Method for the Determination of the Specific Surface of Portland Cement. *Proc. Amer. Soc. Test. Mat.* (1933) **33**, Pt. II, 553-570.

is inserted in the guide and rotated slightly without pressure. The anvil is placed on the column, the ball is placed in the magnet, and released by switching off the current. The crushed material is washed into a test tube with kerosene from the turbidimeter stock. The crushing operation is repeated until four portions have been treated. Some clinkers yield too much or too little fine material from four blows for best results in the turbidimeter. If this situation arises, the duplicate test is made with three or five blows, or in rare cases with six blows. The entire accumu-

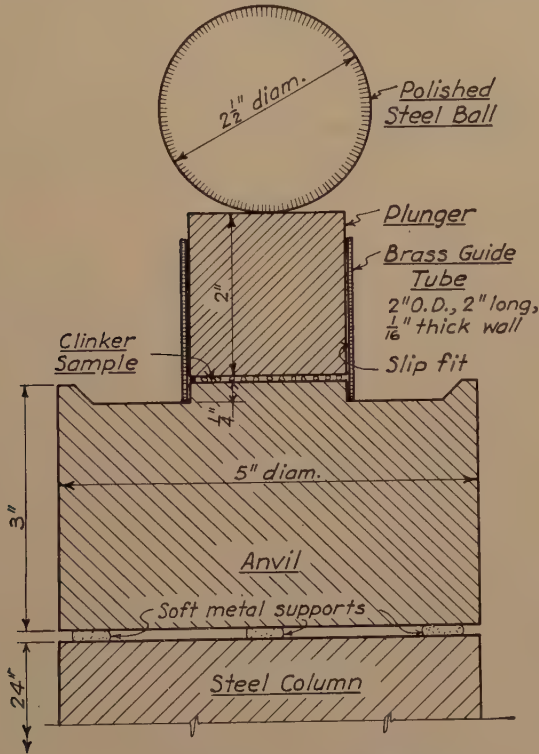


FIG. 1.—APPARATUS FOR IMPACT TEST.

lated product is tested in a Wagner turbidimeter in the manner prescribed for tests of cement in that apparatus.

Reasonably close adherence to the prescribed routine is needed to obtain results in close agreement. The standard procedure was adopted after extensive experimentation, in which test conditions were systematically varied. The results of these studies, which are of value to the student or prospective user of the method, are briefly summarized in the following paragraphs.

Preparation of Sample.—The test is made on a sample of approximately uniform particle size, usually 20 to 28 mesh, cut from a crushed

sample by rough sieving. In this condition, the sample sometimes contains enough dust, mostly finer than 10 microns, to effect a 10 per cent error in results. It is therefore washed with kerosene, rinsed with naphtha and dried by heating. Oversize and undersize material are removed by another sieving. The rotating brush used for dispersing cement in the turbidimeter test is a convenient aid in washing the sample.

Particle Size of Material Tested.—It is desirable to confine tests by this method to clinker samples of nearly uniform particle size. It is not strictly necessary that the test be made on 20 to 28-mesh clinker. Clinker is sufficiently fine grained to warrant the belief that particles of varying size are not inherently different in grindability. In developing this test considerable attention was given to a study of the effect of particle size on the resistance to grinding. So far as reliable experimental work has gone, no great difference has been found in the grindability of different sized particles.

Size of Portion Tested.—The quantity of clinker that can be crushed by a given blow is approximately proportional to the particle size of the clinker. A test on exactly the amount that can be crushed would obviate any inaccuracy due to variable resistance to crushing among the different particles, but would involve tedious experimentation on each clinker. Experiments with varying amounts of clinker show some variation in results, with increasing yields of surface as the quantity treated is increased. It appears that the use of an unduly small quantity is accompanied by increased energy losses, which are not completely measured. The use of quantities greatly in excess of the amount that can be crushed seems to cause a noticeable amount of localized breaking of edges and corners as distinguished from general shattering of the particles. With the general run of clinkers, about one gram of 20 to 28-mesh clinker can be crushed by one blow of the intensity regularly used in the test. The results are in good agreement with those obtained when the amount is selected so as to leave only a trace of uncrushed residue. A 1-gram sample is therefore used in tests of the 20 to 28-mesh fraction. For other fractions or for blows of different intensity the quantity used is varied in proportion to the particle size or the impact energy of the individual blow. Fractions finer than 35 mesh are tested in 0.5-gram portions.

Intensity of Blow.—The selection of the amount of energy to be applied at each blow is a matter of expediency. Inasmuch as the Wagner turbidimeter is used in the test, the amount of fine material in the accumulated product must be such as to give a suitable range of readings on that instrument. This amount of fine material requires the use of a certain amount of energy for its production. This energy can be applied in the form of many light blows or a few heavy ones. Since each blow, whether light or heavy, requires the same manipulation of apparatus and sample, a few heavy blows are used because of the saving of time. A

limiting factor is the increase in energy losses and in the inaccuracy of measuring them as the energy of the individual blow is increased. For the apparatus described, a 24-in. drop of a $2\frac{1}{2}$ -in. ball is a generally satisfactory procedure for tests of fractions coarser than about 35 mesh.

General Technique of Crushing Operation.—The test is made by applying a single blow to each of several portions of dry clinker. In developing the apparatus comparative tests were run on dry clinker and clinker that had been moistened with kerosene prior to crushing. Manipulations were much easier with dry clinker and there was a tendency toward lower surface production with the moistened clinker. The application of repeated blows to a single portion would be desirable from the standpoint of lessened manipulation. This method was investigated and abandoned because of the tendency for the later blows to consolidate the product into hard flakes. As a result the surface-area measurements are less reliable, the energy losses are greater and their measurement is more difficult.

The prevention of dust losses is of considerable importance because even very small losses of fine material are enough to affect turbidimeter readings. The plunger is removed carefully to avoid a rush of air into the space vacated by the plunger. The material is moistened with kerosene and washed into a test tube, and the anvil is scrubbed with a "policeman" after each blow. The plunger and guide are washed and scrubbed after the final blow. The use of excessive quantities of kerosene is to be avoided, so that the entire product can be washed into two test tubes. If more than two tubes are filled, it may be difficult to get all of the sample into the turbidimeter tank.

Calculation of Surface.—The surface area of the material finer than 60 microns in the product is measured by the turbidimeter. The surface of the portion coarser than 60 microns may be computed from a sieve analysis. For this purpose the turbidimeter suspension is decanted through a 325-mesh sieve, and the residue is washed with naphtha, dried and sieved with No. 28, 48, 100 and 200 sieves.

Tests on a large number of clinkers indicate that no serious error is introduced if the total surface is computed by applying a constant factor to the surface indicated by the turbidimeter. The ratio of the surface of the 0 to 60-micron portion to the total new surface varied from 0.82 to 0.86 in tests on more than 20 clinkers. In the method now followed, the surface of the 0 to 60-micron portion is divided by 0.84 to get a value for total new surface. It appears probable that this simplified method introduces no greater uncertainty than exists with very careful hand-sieving and is superior to results that would be obtained with less careful sieving. Omission of the sieve test permits a substantial saving in time.

In order to calculate the surface of the 0 to 60-micron portion, the "transmittancy constant" of the clinker must be known. In tests of

cements this constant is computed from the turbidimeter data and the 325-mesh fineness of the cement. The small amount of fine material in the product of the impact test makes it impossible to obtain a sufficiently accurate 325-mesh sieve determination. Where the test is being made in connection with mill tests, the constant derived on the mill product may be used for the impact test. Where no finely ground product is made, the constant can be evaluated by a turbidimeter test on the fine product obtained by dry-sieving some of the crushed clinker through a 325-mesh sieve, and resieving to insure the absence of coarse particles.

Computation of Energy Applied to Clinker.—The total energy input in the test is the product of the weight of the ball, the distance it falls, and the number of blows. If the anvil is placed directly on the column, the ball bounces on striking the plunger. This bounce represents a loss of energy that is difficult to estimate quantitatively. All or part of this loss may be recovered if the bounce is vertical and the ball returns to the plunger, but sometimes the ball bounces at an angle. The bounce is eliminated if the anvil is allowed to yield slightly at the instant of impact. This slight mobility of the anvil is provided by placing three rods of soft metal between the base and the anvil. The deformation of the rods provides a means of estimating the energy lost in movement of the anvil. This method was used by Gross and Zimmerley, but the heavier anvil in the apparatus described above necessitates the use of a different method of calibration and of softer metal than they used.

The rods used with the apparatus are prepared from $\frac{1}{8}$ -in. solder wire, cut in 0.4-in. lengths and slightly flattened to a thickness of 0.10 to 0.11 in. before use. The procedure for calibrating the rods to obtain their energy-deformation relations is as follows: Three previously flattened rods are placed below the anvil and three or more are placed on the anvil in the position normally occupied by the clinker being crushed. The rods are selected for uniform thickness, are uniformly spaced, and the anvil is placed so as to receive a well centered blow. The thickness of each rod is measured with a micrometer before the first blow and after each of several successive blows. In deriving the energy-deformation relations, it is assumed that the entire impact energy of the ball is divided between the two sets of rods. When the construction of the apparatus and the relative weights of the support column and the anvil are considered, this seems a reasonable assumption. A graphic method is the simplest means of determining the distribution of the total energy between the upper and the lower sets, since deformation is not a linear function of the energy applied. The average total deformations of the upper rods are plotted against the number of blows. The deformation of the lower set after several blows will approximate that of the upper set after the first blow, and the distribution of the energy between the two sets can be read fairly accurately from the curve. From this distribution, the

energy-deformation relations of the rods are calculated. The number of rods in each set must be considered in making the calculations.

With the apparatus described, 85 per cent of the energy was absorbed by the upper set of rods in the calibration test. In a test on clinker, the distribution of energy between the clinker and the rods supporting the anvil may differ from the distribution in the calibration test. In tests of clinker, the energy equivalent of the deformation of the support rods is subtracted from the total energy to obtain the amount of energy applied to the clinker.

The conditions of test affect the magnitude of the energy correction to a considerable degree. The size of the correction increases with increasing intensity of the blow, with decreasing quantity of material and with decreasing particle size. In tests of very fine fractions with a heavy blow, the energy correction becomes large, sometimes exceeding 50 per cent. Tests of the 20 to 28-mesh fraction with blows of approximately 20, 40 and 60 kg-cm. give reasonably consistent results, though the energy corrections vary. Similar results are obtained with light-blow tests on finer fractions, but with a heavy blow even the large indicated correction appears inadequate. There may be some elastic recovery of the solder rods, which becomes noticeable only when deformation is great and rapid. The distance traveled by the plunger in absorbing the kinetic energy of the ball and transmitting it to the clinker varies with the particle size and quantity of sample. The relations between this distance and the time of impact are the probable cause of differences in the amount of the energy correction.

The elasticity of the various parts of the apparatus probably has some bearing on the reliability of the measurements and calibration. As an example, different results were obtained with two different balls of the same weight. A ball described by the manufacturer as high-carbon chrome-alloy steel was used for most of the earlier tests, and was later replaced by a polished steel ball of the type used in ball bearings. The surface production with the first ball was consistently about 12 per cent lower than that with the one used later. A simple bounce test indicated that there was a difference in the elastic properties of the two balls. It thus appears quite possible that this method of estimating energy losses is not highly accurate in an absolute sense. In a relative sense, its precision is such that the difference in net energy in duplicate tests seldom exceeds 3 per cent and averages less than 1.5 per cent.

This calibration procedure is somewhat tedious and even the measurement of the rods before and after the individual test adds appreciably to the time required for manipulation and calculation. In the course of experimentation on a number of clinkers of varying crushing properties, it was observed that the magnitude of the energy correction was practically constant for constant conditions of weight of portion, particle size of clinker, and intensity of blow. As a result of this observation the

present method is to apply a fixed correction of 18 per cent. On the whole, it seems probable that an average correction represents the true situation more accurately than an individual reading.

Calculation of Surface Production per Unit Energy.—The impact test gives an experimentally determined value for surface production per unit of energy input. This value can be variously expressed, but the most usable terms are those which can conveniently be applied to the analysis of mill performance. The unit of production in the cement industry is the barrel of 376 lb. The unit of specific surface is square centimeters per gram. In order to avoid unwieldy values, the surface production for mills is calculated in surface-barrels per kilowatt-hour, obtained by multiplying the increase in specific surface by the output and dividing the product by the mill power. The grindability of clinker is expressed in the same terms. In the apparatus used in these tests the total energy of the blow was 63.4 kg-cm., equivalent to $0.82 \times 63.4 = 52.0$ kg-cm. net energy. If the surface indicated by the turbidimeter test is designated as S_{60} , the total surface is $S_{60} \div 0.84$. One kilowatt-hour is equivalent to 36,710,000 kg-cm., and one surface-barrel is equivalent to $453.6 \times 376 = 170,550$ sq. cm. The grindability value for a four-blow test is therefore:

$$\frac{\text{Surface-barrels}}{\text{Kilowatt-hours}} = \frac{S_{60} \times 36,710,000}{0.84 \times 4 \times 52 \times 170,550} = 1.23S_{60}$$

Accuracy and Precision of Grindability Test.—Experimental errors in the grindability test may occur in either the crushing operation and manipulation or in the measurement of the surface. Experience indicates that duplicate tests on different days should check within 3 per cent for acceptance. The absolute accuracy of the method is dependent on the interpretation rather than on the experimental precision. The systematic errors of surface measurement cancel out if the grindability test is used for the estimation of mill efficiency. The most serious source of doubt is the method of determining the energy actually applied to the clinker. In the absence of adequate theoretical standards, the practical solution is to arrange the crushing operation so as to make highly effective use of the energy, to allow for such energy losses as can be measured, and to accept the results as a working relation between energy used and surface produced. The conditions under which the test is made have been carefully selected so as to balance favorable and unfavorable factors in such a way that the impact test represents what may reasonably be considered as perfect mill performance.

GRINDABILITY TEST IN ANALYSIS OF MILL PERFORMANCE

In conventional mills the production of surface is less efficient than in the impact test. The basic cause of this inefficiency is doubtless the inability of the grinding surfaces to apply their energy effectively to

the particles. Several factors may contribute to this ineffectiveness. The grindability test gives a basis for estimating the degree of effectiveness in the application of energy to the particles being pulverized, by providing means for computing the ideal rate of surface production for a given power input to the mill. The units used in reporting grindability are such that the ratio of the surface-barrels per kilowatt-hour produced in the mill to the grindability value for the clinker is the efficiency of the mill.

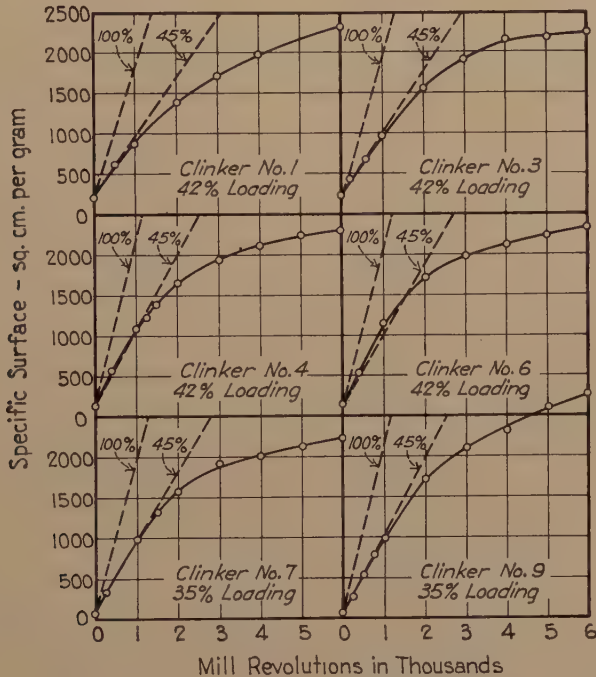


FIG. 2.—GRINDING TESTS IN 29-IN. TUBE MILL.

Laboratory Mill Tests.—In order to get information on the consistency between grindability values and the grinding rate in a conventional mill, it was necessary to grind several clinkers of different grindability in the same mill. The tests were made in a batch tube mill, 29 in. in diameter and 40 in. long. This mill was large enough to permit reasonably reliable measurements of the power input. The net power delivered to the mill was used in the calculations, by making suitable allowances for dead-load losses in the motor and drive. In these batch mill tests, the mill contents were sampled at frequent intervals for surface-area determinations. The results of tests on six clinkers are shown in Fig. 2. In each diagram the solid line is obtained by plotting specific surface against mill revolutions. The positions of the straight dotted lines are computed from the grind-

ability test. The line marked 100 per cent shows the change in surface that would result if the mill were as effective as the impact apparatus in translating mechanical energy into surface production. The line marked 45 per cent shows the change in surface for mill operation 45 per cent as effective as the impact test. The 45 per cent line approximates the actual grinding rate in the earlier stages, indicating a mill efficiency in the neighborhood of 45 per cent.

The consistency with which the impact test indicates the initial tube-mill grinding rate is evidence of the value and essential reliability of the impact test. For these constant mill conditions, the degree of concordance in the instances cited is about as good as the accuracy of the turbidimeter tests and the power readings. Occasional instances have been noted of less exact concordance but there are not enough of them to cast serious doubt on the general value of the impact test. It appears that the grinding resistance of the clinker has no effect on the efficiency of the mill during the earlier stages. As the material in the mill becomes finer, mill efficiency decreases and the rate of surface production drops below the initial rate. This first decrease may be due to the reduction of maximum particle size below the limits of sizes effectively ground by the ball charge. The greater decreases in efficiency at high fineness appear to be caused by ball coating. Clinkers differ in their tendency to coat the balls. This difference may be a specific property of each clinker or may be due to unrecognized differences in test conditions.

The changing rate of surface production is attributed to changing efficiency of the mill. During the earlier stages of the tests in the tube mill, the mill efficiency remains very nearly constant for a time and is very nearly the same with different clinkers. In this part of the tube-mill test, the impact test data are consistent with the tube-mill data. In the later, less efficient stages, no such consistency exists because the reasons for decreased efficiency and the extent of the decrease are varied, and are not necessarily related to grindability.

Commercial Mill Tests.—This method of analysis has been applied to several commercial mills of various types. The design of mills and the commercial requirements imposed on the product limit the test conditions to a narrower range than that covered by the batch mill tests discussed above. At outputs that yield a product coarser than about 1400 specific surface, most of the mills tested operated at an efficiency between 25 and 40 per cent. These values are much higher than have been estimated on theoretical grounds by various investigators. It may reasonably be doubted that the theoretical grounds on which these estimates of extremely low efficiency are based are applicable at all to the pulverizing operation. In view of some of the impressions of extreme inefficiency of pulverizers, the indications of these tests are rather reassuring. The reasons for the spread in efficiency among various mills are not clearly

recognized. The margin between 25 and 40 per cent efficiency is a substantial one, and worth the attention of the designer and operator.

Of necessity, the power figures in the commercial mill tests were for electrical power supplied to the mill motor. Dead load losses, in the motor, the drive and the mill bearings, will account for at least 20 per cent of this total. A smaller amount is used in movement of clinker within the mill, a power use that yields little return because the impacts resulting from it are too light to shatter the particles. The major source of power loss is probably in the form of ineffective contact between the grinding surfaces and the material being pulverized.

In very fine grinding in tube mills, the balls near the discharge end become coated with reconsolidated fine material, the effectiveness of contact is decreased and mill efficiency becomes lower than in coarse grinding. Proper closed-circuit operation can overcome much or all of this difficulty. The methods of analysis used in these tests suggest a means of judging the efficacy of closed-circuit operation by comparing efficiency at high and low output rates.

TABLE 1.—*Mill Efficiency Test Data*

Type of Mill	Power, Kw.	Output, Bbl. per Hr.	Kw.-hr. per Bbl.	Specific Surface, Sq. Cm. per Gram		Surface- Bbl. per Kw.-hr.	Grind- ability of Clinker	Efficiency of Mill, Per Cent
				Feed	Product			
Compartment.....	320	55	5.8	"	1430	246	970	25
Compartment.....	320	42	7.6	"	1700	223	970	23
Compartment.....	320	34	9.4	"	1780	189	970	19
Compartment.....	320	29	11.0	"	1760	160	760	21
Compartment.....	330	82	4.0	"	1410	351	1060	33
Tube.....	370	82	4.5	1410	2040	140	1060	13
Compartment.....	430	104	4.1	"	1540	372	860	43
Ring-roll.....	265	165	1.6	"	540	336	1050	32
Tube.....	310	90	3.4	540	1610	315	1050	30

" Negligible specific surface.

Some representative test data are recorded in Table 1. The tests indicate that there is room for substantial improvement in the design and operation of clinker-grinding equipment. By setting a probable theoretical limit of performance, the grindability test is helpful in estimating the distribution of power losses and in analyzing the possibilities for improvement.

GENERAL DISCUSSION

As a grindability test for Portland cement clinker, the impact test described in this paper has some definite advantages over other methods

that have been used. These advantages lie in the combination of an improved measurement of surface production with a measurement of the energy actually applied to the clinker.

The recent development of the Wagner turbidimeter has made the measurement of surface area rapid and comparatively accurate. This instrument is probably not capable of an extremely high degree of absolute accuracy. The experimental evaluation of the constant relating light absorption to surface is subject to some error. Irregularities of particle shape and translucence of light colored or extremely minute particles introduce inaccuracies in the conventional treatment of the observed data. However, the Wagner instrument provides a measure of surface area that is, for all practical purposes, a close approach to the actual surface area. Moreover, and of particular importance, the systematic errors in surface measurement are of the same sort in both grindability test and mill tests on the same clinker. Thus, from the standpoint of surface measurement alone, greater inconsistencies would be expected among grindability tests on various clinkers than between grindability and mill tests on the same clinker.

The chief virtue of using the impact of a falling ball for the pulverizing operation is that this method permits the relation of surface production directly to energy input. A single impact is applied to each portion of clinker. This feature insures greater constancy in test conditions than is possible in other types of grindability-test devices. In consequence, the results of the impact test are a measure of the pulverizing properties of the clinker, divorced as completely as possible from questions of pulverizer efficiency. There is no theoretical proof that the impact test provides an absolutely complete utilization of energy in the form of pulverizing. However, general principles of mechanics indicate that a close approach to this ideal condition is obtained in the test, and extensive experimentation has apparently eliminated the most serious departures from this condition. The test makes no attempt to do more than measure directly the amount of surface that is produced by applying a given amount of energy. Other effects, such as the production of heat, are not measured. The test measures only the "pulverizing component" of the energy used.

In the course of the development of the impact test for clinker, some observations were made which contributed materially to a better understanding of the mechanism of the pulverizing process. While the tests were confined to clinker, the generalizations based on them probably apply to other close-grained, rigid materials. As a result of these observations, it was concluded that much of the pulverizing action that takes place in conventional mills consists of shattering the coarser particles by impact. When a particle is subjected to an impact of sufficient intensity, it shatters into many pieces, ranging in size from less than one micron up to a top size that depends on the intensity of the blow. Apparently

much of the grinding action in a conventional mill is accomplished by a process of this sort. A blow too light to shatter the particle completely, or a glancing blow, may cause local shattering at the point of contact, or a chipping of corners and edges. Some action of this sort seems to take place during the entire grinding process, becoming more pronounced as the particle size is reduced. A small amount of grinding may occur by a process of attrition or abrasion in which the particles are rubbed by the grinding surface. This is similar to chipping of corners except that the product is finer. Whether reduction of particle size takes place by complete shattering, by localized shattering, or by simple breaking or chipping, the process consists of shearing the particle. For each unit of area subjected to shear, two units of area of new surface are formed. This is the basis of the generally accepted Rittinger law, that surface production is the measure of work done in pulverizing.

The grading of the product of impact is remarkably constant for varying conditions of crushing the same clinker. There are differences in the grading of the products of different clinkers, particularly in the grading below 60 microns as calculated from the turbidimeter test. The specific surface of the 0 to 60-micron part of the impact product ranges roughly from 1000 to 1500 for different clinkers, equivalent to an average particle size ranging approximately from 20 to 12 microns. The product of a conventional pulverizing operation is this impact product, augmented by the extremely fine material made by chipping of edges and localized shattering.

The impact test thus crushes the clinker by a process closely akin to that of the ordinary pulverizing mill. Much of the product of pulverizing in conventional mills is the result of successive shatterings, and all of the new surface results from shearing of the particles. The impact test, therefore, seems a proper and logical basis for establishing standards of performance of conventional pulverizers.

The impact test described here has been developed specifically for Portland cement clinker. The properties of cement are more intimately related to surface area than to sieve fineness. The surface-area basis of analyzing clinker-mill performance is therefore proper, on practical as well as theoretical grounds. Most other materials are pulverized for the primary purpose of reducing all the material to particles smaller than a certain critical or specified size, and surface area is of only secondary concern. A mill grinding such a material might be operating efficiently by the surface-production analysis, and yet be producing much of the new surface from material already fine enough, or by processes other than shattering of the oversize pieces. Judged by the purpose of the operation, this kind of grinding is inefficient. The turbidimeter could probably be adapted to other materials than clinker. If so, the impact test would facilitate the analysis of mill performance on the surface-production basis

and might find some application as a means of estimating the amount of overgrinding of material below the critical size.

As applied to clinker, the impact grindability test has several useful functions. It is useful in the study of the effects of various factors of mill design and operation. It facilitates comparative tests of different mills even when operating on clinkers of diverse grinding characteristics. It gives some idea of the margin of improvement theoretically attainable. In addition to these items, helpful in the analysis of mill performance, the test is the fastest and simplest method available for studying the effect of various treatments of clinker on its grindability. The speed, simplicity and apparent reliability of the impact grindability test recommend it as a useful device for improving the definiteness and precision of mill testing.

Metal Consumption in Hammer Mills at Norris Dam

By FRANCISCO CADENA,* MEMBER A.I.M.E.

(New York Meeting, February, 1937)

THE construction of Norris Dam, built by the Tennessee Valley Authority on the Clinch River, a tributary of the Tennessee River, involved the production of coarse and fine aggregate for approximately 1,000,000 cu. yd. of concrete. A hammer-mill plant for the manufacture of sand from quarried rock was indicated as the most economical method of providing this material. The abrasive character of the feed as compared to the average hammer-mill feed made it necessary to control operations rather closely, and this resulted in the accumulation of many data bearing on production and maintenance. This paper will deal only with three specific sets of data, accompanied by such general information as may be necessary to explain the conditions under which they were obtained and followed by discussions on the relation of silica content to hammer wear, comparative costs of slugger and stirrup hammers, and comparative costs of tool-steel and manganese-steel grate bars.

COARSE AGGREGATE

Aggregate for the concrete poured at Norris Dam came from a dolomite quarry within 2000 ft. of the west abutment of the dam. Geologically this rock is known as a Knox dolomite. A typical chemical analysis is given in Table 1.

TABLE 1.—*Typical Chemical Analysis*

	PER CENT
H ₂ O, 110° F.....	0.02
Ignition loss.....	43.56
MgO.....	19.19
CaO.....	29.07
Fe ₂ O ₃	0.39
Al ₂ O ₃	1.94
SiO ₂	5.75
SO ₂	Tr.

Rock was carried by 12-yd. dump trucks to a 42-in. gyratory crusher set at 6½ in. The product of the primary crusher was taken by a belt conveyor to a double-deck vibrating screen followed by a 5½-ft. cone

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crusher set at 1 in. All the oversize from top deck (+6 in.) went through the secondary crusher. Oversize from lower deck (-6 in. +3 in.), which was the largest size of aggregate used, could either be fed to the crusher or by-passed, depending on the needs for this size of aggregate. The -3 in. material by-passed the crusher. The secondary crusher product and the by-passed material was taken by inclined belt conveyor to the screening structure, where it was separated into the following approximate sizes: -6 +3 in., -3 +1½ in., -1½ +¾ in., -¾ +¼ in. and -¼ in. The +3-in. material went to storage. As much of the next three sizes went to storage as was needed for concrete. Any excess went to the sand plant to be fed to the hammer mills. The -¼ in. material, although already reduced to sand size, contained a large amount of slabs and splinters. It was also passed through the hammer mills, which improved the shape of the particles, making them more cubical.

SAND PLANT

At the sand plant, the -3-in. feed (see Table 2) was crushed in hammer mills to approximately ¼ in. Minus ¼-in. material was split into coarse sand (+8 mesh) and fine sand (-8 mesh). The -8-mesh contained an excess of -100-mesh, which was removed by washing, reducing the quantity from about 32 per cent in feed to washers to about 14 per cent in finished fine sand.

TABLE 2.—*Screen Analyses*

Typical Screen Analysis of Hammer-mill Feed ^a		Composite Screen Analysis Hammer-mill Product ^b	
Size, Inch	Per Cent Through	Size, Inch	Per Cent Through
3 square	1000	¾	100.0
2½	90.2	Mesh	
2	79.5	3	98.9
1½	65.8	4	94.0
1	46.5	8	76.8
¾	35.8	14	56.3
½	32.3	28	41.2
⅜	29.9	48	31.9
Mesh		100	25.3
3	26.5		
4	21.8		
8	13.9		

^a The average of 29 samples taken during March 1935. Variations from month to month were not great.

^b Calculated composite analysis representative of period July 1934 to November 1935. With exception of last two months, when -100-mesh dropped to 20 per cent, variations from month to month were slight.

Hammer Mills.—Hammer-mill installation consisted of four units. Two of these were originally equipped with slugger-type hammers and had adjustable lower breaker plate and adjustable grate bars spaced at 2 in. Feed was introduced from the top along the vertical center line of the mill. The other two units were equipped with stirrup hammers and had fixed breaker plates and fixed grate bars spaced at $1\frac{1}{4}$ in. Feed was introduced on the up-running side of the mill through a gravity chute in a line intersecting the center line of mill and at about a 45° slope. Effective inside dimensions were essentially the same for both mills; i.e., 4 ft. long by 42-in. diameter hammer circle. Each mill was direct-connected to a 250-hp., 880-r.p.m. slip-ring induction motor. During the last few months of operations, stirrup-type hammers were used in both types of mills.

The hammer mills were operated in a closed circuit with six vibrating screens. All of the oversize from top deck ($+1\frac{1}{4}$ -in.) was returned to the hammer mills for further crushing. Oversize from lower deck (+8-mesh) or coarse sand could be either returned to hammer mills together with $+1\frac{1}{4}$ -in. material or sent to coarse-sand storage. Normally, it is estimated that about 40 per cent was returned to the mills. Circulating loads varied from a minimum of about 25 per cent to a maximum of about 100 per cent (based on net output), and averaged about 50 per cent. The circulating load passed a 1-in. square hole. Variations in circulating load were due to variation in the condition of hammers, breaker plates, grate bars, and to some extent to variations in screening efficiency. Since screening was done dry, 8-mesh separation was considerably affected by moisture during rainy weather. The -8-mesh material went to washers, where the excess fines were removed.

Total production from hammer mills for the entire period of operations (July 1934 to April 1936, inclusive) was 878,600 tons.

Control Methods.—Samples for screen analyses were taken at least once a day, and average screen analyses representing at least 30 samples were calculated at the end of every month. All hammers were accurately weighed before and after being used. Tonnages were prorated to the various mills and in turn to hammers, liners and grate bars on a power-consumption basis. The average load on each motor was calculated from recording ammeter charts for every 24-hr. period, and horsepower-hours obtained by multiplying by the net operating hours. Recording ammeter charts also served to keep an exact record of net operating time of each mill and the net operating time that each set of hammers or other wearing parts remained in the mills. At the end of every month, the tonnage produced was estimated from tonnages going through the weighing batchers at the mixer plant. An adjustment was made for either a gain or loss in storage. Since under normal conditions this gain or loss constituted a small percentage of the total production, any error that may have been

made in estimating it would be negligible when applied on the total tonnage for the month. The error on periods longer than one month will be smaller, of course. From the tonnage figure and the total horsepower-hours for the month, the tonnage per horsepower-hour was calculated. This factor was used in distributing the tonnage to the mills and various kinds of hammers used.

RELATION OF SILICA CONTENT TO HAMMER WEAR

In Table 3 are given data on hammer wear for the period November 1934 to November 1935, inclusive. The hammers were manganese-steel

TABLE 3.—*Hammer Wear*

Period of Time	Hammer-metal Wear ^a			Percentage of SiO ₂ in Fine Sand ^b	Average
	Lb. per Ton	Lb. per Hp-hr.	Average		
November, 1934.....	0.034	0.0105	0.010	5.75	5.05
December.....	0.035	0.0100		4.36	
February, ^c 1935.....	0.048	0.0136	0.013	6.46	7.09
March.....	0.046	0.0132		7.68	
April.....	0.045	0.0139		6.65	
May-June.....	0.040	0.0138		8.54	
July.....	0.034	0.0116		6.12	
August.....	0.026	0.0076	0.008	5.59	4.69
September.....	0.022	0.0086		4.74	
October-November.....	0.020	0.0082		3.74	

^a During these tests, 944 hammers were used. Production from these hammers totaled 278,578 tons.

^b Chemical analyses were made on composite sample of fine sand made up of smaller samples taken every 8-hr. shift.

^c January 1935 is omitted from the comparison because no silica determination was made that month.

stirrup hammers used in mills with fixed breaker plates and grate bars. They were all bought from the same manufacturer, of a uniform grade of steel. The rate of wear varied considerably from period to period, and since the screen analyses of the feed and product remained approximately the same, these variations must have been mostly due to changes in the silica content. The rate of wear in pounds per ton was probably also influenced to some extent by inefficient screening during periods of rainy weather, when considerable amounts of fines were returned with the circulating load.

The period under consideration falls into three groups: (1) November and December 1934, which may be considered a period of medium silica

content, averaging 5.05 per cent with a corresponding rate of wear of 0.01 lb. per horsepower-hour; (2) February to July, inclusive, a period of high silica averaging 7.09 per cent with corresponding rate of wear of 0.013 lb. per horsepower-hour; (3) August to November inclusive, a period of low silica averaging 4.69 per cent with a corresponding rate of wear of 0.008 lb. per horsepower-hour. (Fig. 1.)

A set of eight hammers (averaging 27.5 lb. per hammer) weighs 220 lb. About 88 lb. (40 per cent) can be worn off the hammers before it is necessary to replace them. This corresponds to 8800 hp-hr. at a rate

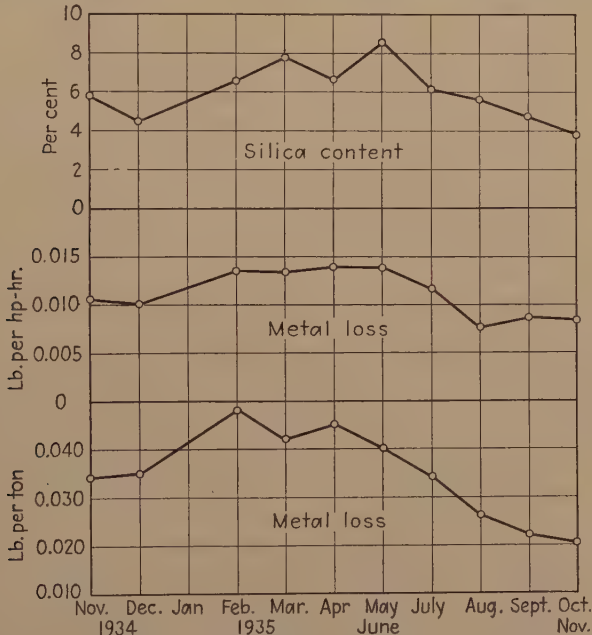


FIG. 1.—SILICA CONTENT AND LOSS OF HAMMER METAL.

of wear of 0.01 lb. per hp-hr. With an average load on motors of 200 hp., the life of a set of hammers would be 44 hr. Since the hammers have to be turned at about $\frac{1}{2}$ the cycle, mills had to be opened every 22 hr. of net operating time when feed had a medium silica content. With low-silica feed, the intervals were about 27 hr. and with high-silica, approximately 17 hr. Approximately $\frac{3}{4}$ hr. was lost in changing or turning a set of hammers, and some additional time was lost in inspections to make certain that hammers did not stay in the mill too long. It is thus apparent that silica contents appreciably higher than 7 per cent would have meant frequent interruptions in the operation of each unit with consequent loss in output per mill.

Suitability of Hammer Mills for Abrasive Feeds.—A silica content higher than 7 per cent would not necessarily eliminate hammer mills from con-

sideration. Coarser crushing would allow for a higher tolerance. Brittle material, which makes for the production of a large amount of fines in a single pass, should also materially increase the tolerance. If the silica present happens to be chemically combined, its influence disappears and the abrasiveness of the mineral involved would be the factor to consider. Although apparently most of the silica present in the Knox dolomite was in the form of chert, it is possible that some may have been chemically combined with alumina. If this were true, rates of wear discussed above would have been somewhat higher if the percentages of silica given had applied only to free silica.

In considering the suitability of hammer mills, the particular application of the product is of prime importance. If, for instance, it is intended for concrete aggregate, the relatively cubical shape of particles produced will justify the use of hammer mills for feeds relatively high in silica. The higher degree of workability in concrete mixes caused by using hammer-mill sand effected a substantial saving in cement at Norris Dam—a saving that fully justified its use.

COMPARISON OF SLUGGER AND STIRRUP HAMMERS

The slugger type of hammer, made in one piece (Fig. 2) is essentially a bar of rectangular cross section, with a shank that fits in between the rotor disks. An entire row (14 hammers) is held in place by a long pin extending the full length of the rotor. The pin is secured against side movement by cotters at the ends. The head of the hammer is 5 in. long (radial dimension), 3 in. wide (dimension parallel to rotor shaft), and 4 in. deep (tangential dimension).

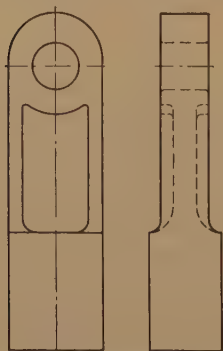


FIG. 2.—SLUGGER HAMMER.

The stirrup hammer consists of two arms and the hammer itself. The assembly of hammer and arms forms a U, or stirrup shape. The two arms slide in between the disks and are held in place by individual pins. Since hammers are placed end to end, four hammers in a row take the place of 14 slugger hammers. Stirrup hammers were $11\frac{1}{2}$ in. long (dimension parallel to rotor shaft), with a tangential dimension of 4 in. and a radial dimension of $3\frac{3}{8}$ in. on most of the hammers used. The terms "radial" and "tangential" refer to the circle described by the revolving hammers.

Figs. 3 and 4 show the two types of stirrup hammers used at Norris Dam. In the pin type (Fig. 3) the arms are secured to the hammer by means of a pin. In the lug type (Fig. 4), one end of each arm slides under the lug. When the other ends of the arms are inserted between

the rotor disks, the assembly is securely locked. Both types of stirrups are held on the rotor by individual pins.

Slugger hammers were turned three or four times. Lower breaker plate and grate bars were adjusted to make up for shrinkage of hammer circle as the hammers were worn. About 4 in. was worn off the head out of a possible 5 in. (Fig. 5.) Stirrup hammers were turned only once (Fig. 6). Approximately nine sets of hammers were used to one set of arms. The cost of arms has been added in comparisons made in Table 4.

The mills equipped with slug-gers were started in December 1934. From the start, hammer costs were considerably higher than in the mills equipped with stirrup hammers. Table 4 shows the comparative data for the two types of hammers. Objection might be raised to this comparison on the ground that distribution might not be fair, since this was made on the basis of power consumed, which assumes equal efficiencies for the mills. It might be argued that the efficiency of the mill with slugger hammers might be higher than that of the other mill. However, tonnage per horsepower-hour was higher when stirrup hammers were being used exclusively before the start of machines equipped with slugger

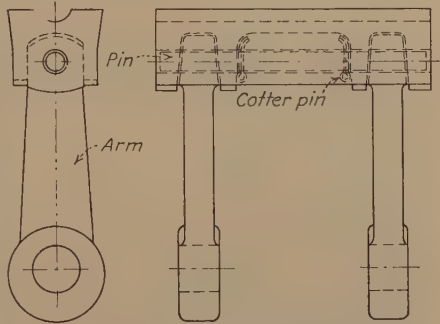


FIG. 3.—STIRRUP HAMMER, PIN TYPE, WITH ARMS ATTACHED.

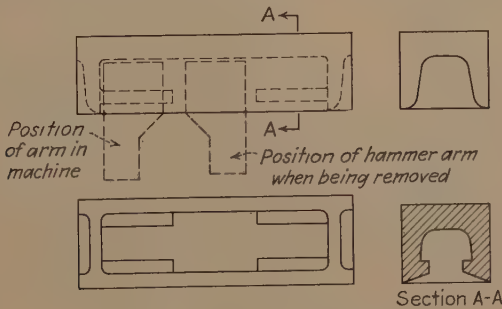


FIG. 4.—STIRRUP HAMMER, LUG TYPE.

hammers. Tonnage per horse-power-hour was also higher after the use of slugger hammers was discontinued.

Another objection might be that since the hammers under comparison are being used under entirely dissimilar conditions as to type of mill, number of rows per set, grate-bar opening, etc., the results obtained cannot be relied upon. The answer to this objection is that the change to stirrup hammers in the machines that had been originally equipped

with sluggers eliminated the difference in hammer costs between the two types of mills.

The comparison made in Table 4 covers the period from January to July 1935, inclusive—a period of high silica. Both types of hammers were of the same grade of manganese steel, bought from the same manufacturer.

It should be borne in mind that under conditions different from those obtaining at Norris Dam it is possible that slugger hammers may be more economical. It might be mentioned also that by making a slugger hammer in two pieces—a shank and a renewable tip—costs comparable to those of stirrup hammers might have been possible at the Norris installation.

TABLE 4.—*Comparison of Stirrup and Slugger Hammers*
JANUARY TO JULY 1935, INCLUSIVE
Distributions Made in Proportion to Power Consumed

	Stirrup Hammer in Mill with Fixed Breaker Plates and Grate Bars. Grate- bar Spacing, 1¼ In. ^a	Slugger Hammer in Mill with Adjustable Breaker Plates and Grate Bars. Grate-bar Spac- ing, 2 In. ^b
Number of hammers per row.....	4	14
Number of rows.....	2	3
Number of hammers per set.....	8	42
Total number of hammers used.....	745	334
Total power consumption, hp-hr.....	652,578	177,929
Total tonnage.....	201,687	54,992
Average load on motors, hp.....	208.1	208.3
Tons per horsepower-hour.....	0.30906	0.30906
Horsepower-hours per hammer.....	875.9	532.7
Tons per hammer.....	270.7	164.6
Average weight of new hammers, lb.....	27.5	30.0
Average weight of used hammers, lb.....	16.2	18.0
Average loss of weight per hammer, lb.....	11.3	12.1
Total weight of new hammers, lb.....	20,502	10,039
Total weight of used hammers, lb.....	12,040	6,023
Loss of weight, lb.....	8,462	4,016
Loss of weight, per cent.....	41.3	40.0
Loss of weight, lb. per hp-hr.....	0.013	0.023
Loss of weight, lb. per ton.....	0.042	0.073
Worn metal and discarded metal, lb. per ton.....	0.102	0.182
Cents per ton (@ 14.5¢ per lb. for metal).....	1.48	2.64
Cents per ton with 0.17¢ added for stirrup-hammer arms.....	1.65	2.64

^a Mill with fixed breaker plate and grate bars gave a finer product. This is due, at least partly, to closer spacing of grate bars.

^b Adjustable grate bars and breaker plates were kept as close to the hammers as possible.

COMPARISON OF TOOL-STEEL AND MANGANESE-STEEL GRATE BARS

The comparative data given in Table 5 are based on the service obtained from the two kinds of grate bars that were used in the mills with fixed breaker plates and fixed grate bars. All conditions, as far as could be ascertained, were the same except that tool-steel bars were used in one and manganese grate bars in the other. Comparison covers the period from March to November 1935, inclusive. Silica was high at the beginning of the period and low at the end, which makes it more or less representative of conditions for the entire duration of operations at Norris. It may be added, however, that the actual cost of tool-steel grate bars was higher than that obtained during the period of comparison covered by Table 5. This higher cost was due to breakage during the early stages of



FIG. 5.—WORN SLUGGER HAMMER.



FIG. 6.—WORN STIRRUP HAMMERS.

Top, lug type; bottom, pin type.

operations and also to closer spacing in the first few sets used. Elimination of tramp iron by installation of lifting magnets ahead of hammer mills, increasing the thickness of bars from $\frac{1}{2}$ to 1 in. and increasing the spacing from $\frac{1}{2}$ to $1\frac{1}{4}$ in. reduced the costs. These reduced costs were still much higher than for cast manganese bars (Table 5). Each set of grate bars consisted of 12 sections. Dimensions were essentially the same for both kinds. There was practically no breakage of tool-

steel grates during the period under comparison and none whatever in the manganese bars.

TABLE 5.—*Tool Steel versus Manganese Steel for Grate Bars*^a

Tool Steel			Cast Manganese Steel		
Tons Produced	Cost per Ton		Tons Produced	Cost per Ton	
	Dollars	Cents		Dollars	Cents
119,453	2,171.16	1.82	133,880	686.40	0.51

^a During the period of the test, 6 sets of tool steel and 4¼ sets of cast manganese grates were consumed.

DISCUSSION

(*Paul M. Tyler presiding*)

H. J. BROWN,* West Newton, Mass.—I think we are mixed in our nomenclature; what is the real distinction between grinding and fine crushing? Could we not define grinding as reducing to micron sizes, and crushing as reducing to sieve sizes?

S. B. KANOWITZ,* New York, N. Y.—Crushing usually relates to production of materials having rather narrow limits between maximum and minimum sizes, while grinding relates to production of materials with an upper size limit only; that is, a great deal of very fine materials may be present.

* Consulting Engineer.

Flotation Processing of Limestone

By BENJAMIN L. MILLER,* MEMBER A.I.M.E., AND CHARLES H. BREERWOOD†

(New York Meeting, February, 1935)

FROM earliest recorded times, limestone has been employed in the industrial life of peoples of all sections of the world where it exists. It is widely distributed and therefore has been available in almost every country and has been employed for a variety of purposes. Its earliest use was for buildings, since it is more easily quarried and dressed than most other kinds of rocks and so could be utilized by early inhabitants, poorly equipped with tools. The next use was in the manufacture of lime, which was employed by the ancients in mortar. These two uses of limestone are still all important, but they have been overshadowed in scores of regions by literally hundreds of new applications for raw limestone or products made from it. Anyone familiar with modern manufacturing processes can quickly call to mind many industries in which limestones are essential, and new uses are continually coming to notice. Any enumeration would probably be incomplete and soon out of date.

ORIGIN OF LIMESTONE

Disregarding details and unusual types concerning which often there is insufficient evidence and considerable difference of opinion, limestones have been formed by the precipitation of calcium carbonate from solution through the action of organisms, both plants and animals, or in a limited number of instances by chemical reactions without the intervention of living forms. Some of the organisms extracted the calcareous material in order to build up hard structures, either as bony skeletons or as coverings (clams, snails, etc.), whereas others in their life functions emitted substances that resulted in the precipitation of the soluble constituents outside the organisms themselves. In some instances the organisms utilized only calcium carbonate but in some cases other compounds as well were precipitated. Various materials have been found in the hard parts of animals, and for that reason it is almost impossible to find a pure natural limestone; that is, one composed entirely of CaCO_3 .

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The places where limestones have been formed also must be considered in explaining their constitution. Although there are a number of environments where calcareous precipitation may take place, geologists are now in accord in their belief that the great majority of limestones have been formed in shallow marine waters where animal and plant life is abundant, although some have been formed in fresh waters. The great majority have been accumulated as oozes or shell deposits in close proximity to land. Limestones are now in process of formation and their environmental conditions have been investigated.

The impurities of limestones are of the greatest economic importance, and their sources should be recognized. Most of them are derived from previously existing rocks of land areas. With the agencies of transportation such as gravity, wind and running water, it is only reasonable to expect that some of the products of rock decomposition and land erosion should be carried in the form of mud or sand into the bodies of water where the limestones were in process of accumulation. Even remote from the shores, impurities in the form of dust or fine sand may be dropped by the wind or floated out from the shores where they finally settle to the bottom. These land-derived materials may include practically all the resistant minerals of all classes of rocks as well as carbonaceous matter of plants.

In addition to the primary causes for limestone impurities, there are numerous ways by which the original constitution may be greatly modified. Circulating underground waters affect the constitution of limestones both by the solution and removal of some of the soluble constituents and by the introduction of foreign materials. When rocks have suffered severe compression during great earth movements, heated waters have circulated freely through the shattered rocks. Removal of part of the very soluble calcium carbonate and the addition of foreign minerals increases the percentage composition of the impurities and changes the character of the rock.

It is therefore a matter of surprise rather than otherwise that limestones are as pure as they are, with so many ways in which they may receive contamination. On casual examination, freshly broken limestones may lead one to believe that they are fairly pure and uniform throughout. Weathered surfaces, however, commonly reveal their heterogeneous character, and the polarizing microscope brings out still better the lack of homogeneity. The irregular distribution of the foreign minerals and the varying amounts in different strata and even within the same bed are striking.

MINERAL COMPOSITION OF LIMESTONE

The mineral impurities of limestones are many and varied; some of them making the stone more desirable for certain purposes but at the

same time unfitting it for other specific uses. A discussion of all these products cannot be given here. Instead, attention will be directed to only a few that are of major importance.

Quartz in the form of small sand grains, in thin veins or in irregular particles filling pore spaces in the rock, is one of the most common impurities of limestones. For many limestone uses, quartz is beneficial, but in most manufacturing processes where lime or limestone is employed, it is a disadvantage. Other forms of silica, such as flint, chert, chalcedony, etc., are also apt to be present.

Silicate minerals are commonly present, especially micas (aluminum silicates with varying amounts of potassium, sodium, magnesium, iron, etc.), the ingredients of which were transported from the land as mud. Feldspars (potassium and sodium aluminum silicates) and ferromagnesian minerals (hornblende, pyroxene, etc.) are often noted.

Magnesium carbonate is almost always present, as many of the organisms were responsible for the precipitation of both calcium and magnesium carbonates. In addition, underground waters may introduce some magnesium compounds. The percentage of MgCO_3 in a limestone may vary from 0 to 45.61 per cent, the amount present in the mineral dolomite.

Iron, either in the form of oxides (limonite, hematite, etc.), sulfides (pyrite, marcasite) or carbonates (siderite, ankerite), is almost universally present in varying amounts.

Carbonaceous or bituminous matter is present in many limestones. It is derived mainly from plants growing in the ocean waters where the calcareous oozes were accumulating or washed in from the land. When the limestones have been subjected to metamorphic forces, these hydrocarbons may be converted into graphite.

Calcium phosphate is an impurity which locally is present in objectionably large quantities.

Numerous other minerals present are of little economic importance. In the metamorphosed limestones a wide range of oxides, sulfides and silicates have been observed in addition to those specifically named.

USE AND SELECTION OF LIMESTONES

Both the physical and chemical characteristics of limestones are important in determining the uses to which they may be put. For certain purposes a wide variety may be equally satisfactory, whereas for others an extremely definite composition is demanded. The geologist frequently is asked to locate beds of stone possessing certain desirable features and free from certain other undesirable qualities. This he can generally do in a rough way by field examinations, but it remains for the chemist to determine the exact chemical composition and for the petrographer, by the aid of the polarizing microscope, to determine the definite

minerals present. Many of the constituent mineral particles are entirely too small to be recognized otherwise.

For the gross uses of limestones, the refinements of the chemist and petrographer are not needed, but in the various chemical industries where limestone or lime is used a knowledge of the exact constitution is highly important. The chemist was early called in to assist the cement manufacturer, and his connection became more essential as more rigid specifications for the different grades of cement were formulated. The mineralogical character of limestone has received little attention but it is now apparent that this is almost equally important. In most cases in chemical manufactures, limestone is added for the purpose of obtaining new chemical combinations although it is also valuable as a flux. The chemical analysis may reveal the necessary elements for the manufacture of the desired product but it does not necessarily determine the chemical recombinations that may result under the temperature and other environmental conditions of the manufacturing process. The mineral constitution is of greater significance, because some minerals readily break up and enter into the new combinations desired while others are decomposed only with difficulty. When the cement chemist mixes rocks of different composition in the right proportions to produce the desired tricalcium and dicalcium silicates and tricalcium aluminate, and finds that the burned product is other than expected, and that free lime and free silica are present, he is apt to blame the burning for the failure. Sometimes a second burning is made, and even then free lime is present, although in smaller amounts.

The explanation for unsatisfactory results when the chemical proportions are correct depends on the physical and mineralogical character of the mix. Even with this situation fully recognized, the problem for limestone users has been how to control the mineralogical composition of the limestone. The chief difficulty is the fineness of the constituent minerals, both the desirable as well as the undesirable ones. Until now their separation has seemed to be economically an insurmountable obstacle, because of the low selling price of most of the manufactured articles into which limestone enters.

Another difficulty encountered in cement manufacturing has been due to the variable composition of the limestones as well as of the shales or other substances used. Occasionally a limestone may be found of practically the exact composition needed. More frequently, however, the main stone used is deficient in some chemical elements and it is necessary to obtain another stone for blending. This situation is common throughout the Lehigh Valley, as well as elsewhere. The cement quarry furnishes stone somewhat deficient in lime, and to rectify this defect it may be necessary to purchase high-grade limestone from another quarry or another region. While the lime proportions may thus be satisfied, other ratios, such as that of the silica to the iron-alumina, may

be upset. Making use of raw limestones as they are found in nature seldom permits a cement manufacturer to put into the kilns a mix that in all respects meets with his wishes. In other words, cement manufacturers generally have not been able to produce the best cement possible, because they have been obliged to use the stones as they have come from the quarry.

With the demand for special cements for special purposes, such as "early strength," "low heat producing," "white," etc., or with reference to composition "high silica," "low alumina," "high alumina," "high iron," etc., more and more difficulty has been experienced. If the manufacturer is compelled to use stone as it comes from the quarry, only certain cements can be produced in one region, since transportation costs prohibit the shipment of the raw materials great distances.

With these inherent difficulties confronting the cement manufacturer, as well as other users of limestone, it has become more and more important to separate the stones into their mineralogical constituents and then re-combine them in proper proportions for each purpose. Until recently, this procedure seemed impracticable. Fine crushing and screening was tried and some results of consequence were secured, but of no practical significance; for example, it was found that the major portion of certain minerals, particularly quartz and to a lesser degree mica, were not reduced to as fine a size as the calcite during the grinding process.

Gravity separations on shaking tables were made, but with indifferent success because of the slight differences in the specific gravity of the more abundant component minerals.

With the arrival of flotation in the field of the nonmetallics, it seemed worth while to attempt the processing of limestone by froth flotation. Experiments conducted by C. H. Breerwood have met with success, and on March 15, 1934, the flotation plant of the Valley Forge Cement Co. was put in operation. The description that follows is almost entirely concerned with that installation.

PROCESSING AT VALLEY FORGE

The Valley Forge Cement Co., in the eastern part of the Chester Valley, at West Conshohocken, Pa., uses a limestone of composition similar to that so extensively worked in the Lehigh Valley although of very different physical appearance. It has been named the Conestoga limestone by the geologists of the U.S. Geological Survey, but should more properly have been called the Jacksonburg, as it was formed at the same time as the Jacksonburg limestone of the Lehigh Valley, and under similar conditions. The two areas doubtless were at one time continuous but, owing to earth movements and subsequent erosion, are now separated by a distance of about 40 miles. The rocks of the Chester Valley were subjected to much more intense compression than those of the Lehigh Valley during the two extensive periods of mountain building

that have taken place in eastern Pennsylvania since their deposition; therefore they have undergone much greater metamorphism.

The Valley Forge rock when broken presents an appearance of mica schist on the bedding-plane surfaces, owing to the development of fairly coarse overlapping flakes of sericite mica along the bedding planes.

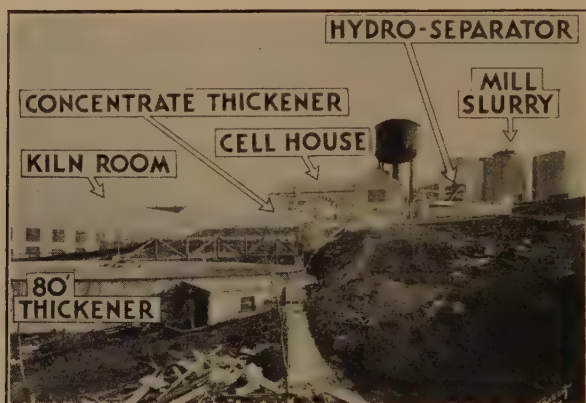


FIG. 1.—GENERAL VIEW OF FLOTATION PLANT OF VALLEY FORGE CEMENT CO., WEST CONSHOHOCKEN, PA.



FIG. 2.—INTERIOR VIEW OF CELL HOUSE, VALLEY FORGE CEMENT CO.

The broken edges reveal coarse crystals of gray to white calcite. The peculiar appearance of the rock led to its rejection for cement manufacture by people who examined the region before the building of the plant of the Valley Forge Cement Co. All of the minerals of the rock occur in coarser particles than in most cement rock. Experimentation showed that the coarse flakes of mica decomposed at the heat of the kiln and recombined in the same manner as the ingredients of the finer grained rocks of other regions.

On the company's property there is a large amount of stone that contains sufficient lime for the manufacture of portland cement without the addition of any other stone. The property has been self-supporting since its opening in 1927. However, there is an enormous amount of other stone on the property that is too low in lime. Some care has had to be taken in the operation of the quarry to avoid this low-grade stone. Also, it has been necessary to carefully remove and waste the surficial stone from which much of the calcareous material has been removed. This is a common practice in most cement districts. The quarrying expense has been somewhat increased by this necessity.

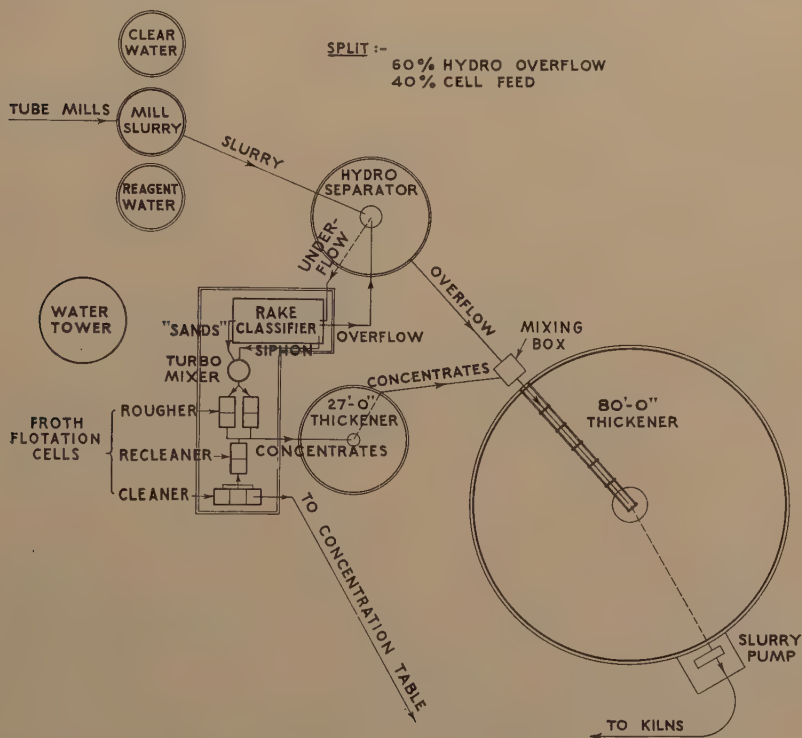


FIG. 3.—FLOW SHEET OF FLOTATION PLANT, VALLEY FORGE CEMENT CO.

The flotation plant of the company has been described in recent publications¹, so that the following descriptions are brief. The flow sheets are shown in Figs. 3 and 4.

¹ Patent No. 1931921: Manufacture of Cement. U.S. Patent Office, Washington, 1933.

N. C. Rockwood: Chemistry Applied to Cement Manufacture. *Rock Products* (August, 1934) **37**, 32-37.

C. H. Breerwood: Mineral Separation Applied to Cement Manufacture. Conservation Committee, Portland Cement Assn., Philadelphia, 1934.

Chemical Control Conquers Cement. *Chem. Ind.* (1934) **35**, 412-414.

Grinding

The first stage in the processing is to grind the rock to a fineness sufficient to separate most of the mineral particles. Obviously it is impracticable to grind to the point where all the bonds are broken, as some calcite will be cemented to quartz or mica even in the most minute sizes. As the result of experiments on the Valley Forge stone, it has been found that it is sufficient to grind so that 85 per cent passes through a 200-mesh screen. After the removal of the coarse pieces of quartz and mica by flotation, it has been found that 91 to 92 per cent is less than 200-mesh.

At the plant the grinding is carried on in a one-stage wet process. It is believed that grinding costs could be reduced by grinding to a point where 65 per cent passes through the 200-mesh screen and then separating the coarse quartz by flotation and regrinding the concentrates to the desired fineness. The coarser grinding would free much of the quartz and mica, but, naturally, less than when ground finer. Since quartz is hard to grind, there would be a saving in not reducing the useless quartz to the finer size.

At the Valley Forge Cement Co. plant, wet grinding is employed. In the process it is probable that a film of hydroxide is formed about each calcite particle, thus permitting collection by the oleic acid. If the stone is ground dry, agitation in water is necessary in order to wet the individual grains.

Separation of Fines

Investigation has shown that the finest material generally is somewhat higher in lime than the coarser, although all of the impurities are represented in even the finest sizes. The fine quartz and mica particles are in excellent form for recombination at the heat of the kiln, therefore in practice there is a separation between the material finer than 325-mesh from that coarser. This separation is effected by a 30-ft. Dorr hydroseparator and a Dorr rake classifier. The slurry from the tube mills is mixed with water in the hydroseparator to a 20 per cent dilution. The overflow from the hydroseparator goes to the 80-ft. Dorr thickener as part of the finished product. The underflow goes to the rake classifier. From this machine the overflow goes back to the hydroseparator, and the coarser material to the flotation cells.

The hydroseparator is adjusted to regulate the amount of material to be treated by the flotation plant. This is modified according to the composition of the slurry. On the basis of 700 tons of material per day, rock of different grade must be treated as shown in Table 1.

Flotation

The coarse material from the rake classifier passes into the turbomixer. In addition, a portion of the overflow of the rake classifier is taken to the mixer. In Fig. 4 this is labeled "withdrawal." The exact amount of the classifier withdrawal depends upon the composition of the original slurry. If the rock is low in lime, a larger amount is sent to the cells than if it is high. In the mixer the material is diluted to the proper degree and thoroughly mixed for the cell.

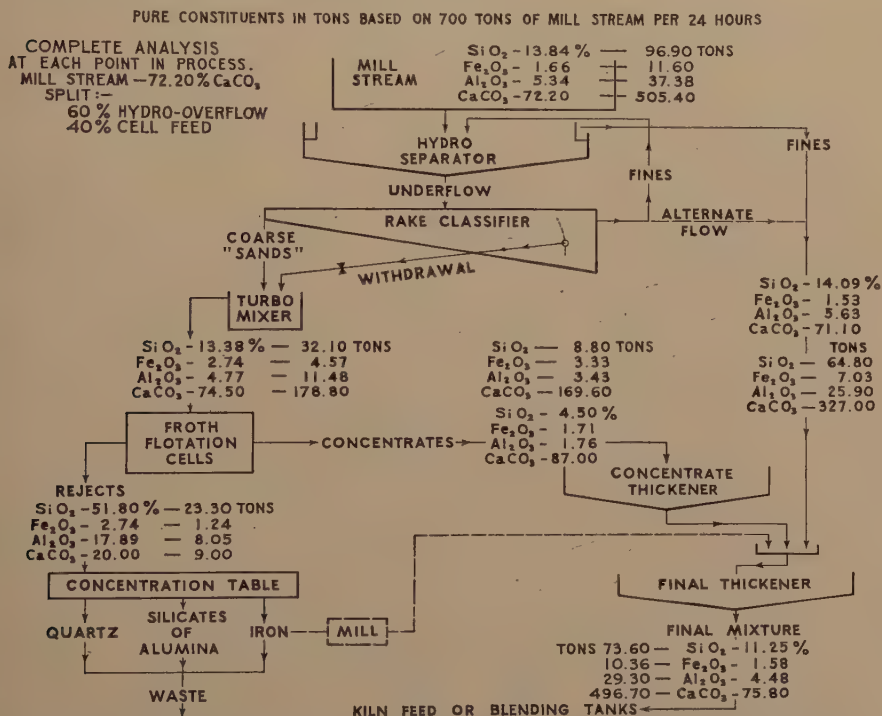


FIG. 4.—FLOW SHEET OF VALLEY FORGE CEMENT CO., SHOWING RESULTS OF ACTUAL OPERATIONS.

As the pulp comes from the mixer, oleic acid, which acts as a collecting agent, is added. The material goes to four rougher cells where cresylic acid is introduced to produce frothing. The overflow from these rougher cells, constituting the first concentrate, is discharged into a 27-ft. concentrate thickener and the underflow, in which there is still considerable calcite, passes through another set of three flotation cells, called the cleaner cells. The overflow from the cleaner cells carries more silica and aluminum silicates than the overflow of the first set of cells, and it is advisable to further treat these concentrates. This is done in two recleaner cells. The underflow from the cleaner cells constitutes the

reject, composed mainly of quartz, mica and iron oxides together with some calcite.

The concentrates from the rougher and recleaner cells go together into the concentrates thickener, in order to separate part of the reagent water. This water is pumped into a special tank for mixing with the new slurry and in this way an economy of reagents is effected. From this concentrates thickener the material joins the stream from the hydroseparator in a junction box, in order properly to blend, then the combined streams pass into the large thickener. A Wilfley pump conveys the combined products to the kilns.

On the basis of the composition of the raw rock, the classifiers and frothing cells would divide an original 700 tons at different stages of the operation into the quantities and compositions shown in Table 1. This table was constructed to show the rejection of material carrying 20 per cent CaCO_3 ; the concentrates contain 87 per cent CaCO_3 . Naturally, in actual operation there will be slight changes in these two figures, which also will affect the other results. The dilutions vary at different parts of the process (Table 2). On the basis of such analyses as are given in the flow sheet, 700 tons would be separated into the tonnages listed in Table 3, at different stages of the concentrating process.

TABLE 2.—Percentage of Dry Solids in Different Parts of System

	PER CENT		PER CENT
Slurry.....	67.0	Cell concentrates.....	16.5
Hydroseparator overflow.....	16.6	Underflow, concentrates thickener.....	59.2
Hydroseparator underflow.....	55.0	Feed of large thickener.....	27.4
Coarse sands from rake classifier....	75.0	Final mixture.....	66.6
Rake-classifier withdrawal.....	43.0	Rejects from cells.....	17.1
Cell feed.....	33.3		

TABLE 3.—Tons of Dry Solids at Different Stages in Treatment of 700 Tons of Slurry

72.2 Per Cent Calcium Carbonate in Raw Stone

Solids	Mill Stream	Hydro Overflow	Cell Feed	Concentrates	Rejects	Final Mixture
Total tonnage.....	700.00	460.00	240.00	195.00	45.00	655.00
CaCO_3	505.40	327.00	178.80	169.60	9.00	496.70
SiO_2	96.90	64.80	32.10	8.80	23.30	73.60
Fe_2O_3	11.60	7.03	4.57	3.33	1.24	10.36
Al_2O_3	37.38	25.90	11.48	3.43	8.05	29.30
MgCO_3	46.40	31.10	15.30	13.40	1.90	44.50

Of course, all of these separations do not actually take place. For example, much of the Al_2O_3 is combined with SiO_2 , and the MgCO_3 is not separated from the CaCO_3 . Nevertheless, Table 3 is valuable for comparison. Perhaps the most striking feature to be noted is the small

amount of CaCO_3 discarded in the rejects. As shown in the typical flow sheet (Fig. 4), the waste, amounting to 45 tons, contains 20 per cent CaCO_3 . Thus in the treatment of 700 tons there is a loss of only 9 tons of usable material, an amount that is insignificant and completely overshadowed by economies effected. Although probably not economical in most cases, it would be possible to recover a large part of this discarded CaCO_3 .

It is important to know how uniform a product can be made by this process. The whole operation is extremely flexible and permits changes at many points, so that, regardless of the changing composition of the quarry rock and the resulting slurry, it is possible to maintain a sufficiently uniform finished product. This is done mainly through the regulation of the amount of material sent to the flotation cells. In a cement plant, little further correction or blending is necessary. As an example of the uniformity of the final processing product, Table 4 is presented.

TABLE 4.—*Daily Composition of Product of Valley Forge Cement Company*

Date, 1934	CaCO_3 from 80-ft. Thickener, Per Cent	Date, 1934	CaCO_3 from 80-ft. Thickener, Per Cent
May 1	76.03	May 17	76.32
2	75.4	18	76.3
3	75.0	19	75.72
4	73.7	20	76.48
5	74.08	21	76.78
6	74.4	22	77.53
7	74.47	23	77.6
8	75.0	24	77.5
9	75.68	25	77.11
10	75.9	26	76.6
11	76.1	27	76.6
12	76.4	28	76.8
13	76.2	29	77.3
14	75.83	30	76.65
15	75.58	31	77.45
16	75.95		
			Average 76.08

Processing of Rejects

So far, the Valley Forge Cement Co. has made no use of the rejects from the flotation cells. Experimentation on shaking tables has shown the possibility of separating the iron compounds, the quartz and the micas. Analysis of the material as put on the table, and the products obtained, are as follows: SiO_2 , 64.46 per cent; Fe_2O_3 , 2.72; Al_2O_3 , 13.68; CaO , 6.02; MgO , 3.16. The feed of this composition was separated into five divisions, as shown in Table 5.

TABLE 5.—*Division of Feed*

	Concentrate End of Table, Per Cent				Side of Table, Slimes, Per Cent
	No. 1	No. 2	No. 3	No. 4	
SiO ₂	2.60	57.40	74.60	75.78	58.50
Fe ₂ O ₃	59.10	2.43	1.23	1.44	1.14
Al ₂ O ₃	9.86	7.05	7.31	9.56	22.00
CaO.....	7.53	16.36	7.22	4.41	3.21
MgO.....	1.46	1.60	0.95	0.59	4.24
Loss.....		11.83	3.02	3.25	4.85

The results show the separation of some materials that might be utilized in cement manufacture as well as for other special purposes. For example, concentrate No. 1, with a high iron content, might be used in the manufacture of a high-iron cement. Nos. 2, 3 and 4, which together constitute 38 per cent of the feed, with an average silica-alumina ratio of 8 to 1, could be used to correct low ratios. If these materials were to be used, they would first need to be reground, since many of the particles are larger than 325 mesh.

Many uses for the rejects have been suggested, and it seems probable that further experimentation will be carried on, with the ultimate result that very small amounts, if any, of the materials separated from the limestones will be wasted.

Mineral Composition of Flotation Products

In the foregoing discussion little mention has been made of the mineral composition of the products at different places in the process. From the raw rock to the final concentrates and rejects, mineralogical examinations were made of the various materials by means of the petrographic microscope, by Dr. Donald M. Fraser. The minerals determined were the calcium carbonate minerals (calcite and dolomite), the aluminum silicates (mainly muscovite mica), quartz, limonite, pyrite, carbon (principally amorphous), amorphous silica and apatite. Dolomite has not been mentioned in the processing descriptions, although almost universally present in small amounts. It is generally intimately mixed with the calcite and not separated from it. The two minerals have been termed "calcite" in all the operations. Quartz, mica and calcite (including dolomite) constitute the bulk of the rock, although in some specimens there is considerable limonite and carbonaceous matter with subordinate amounts of the other minerals.

In the processing there was little difference in the mineral composition of the slurry and the underflow and overflow of the hydroseparator. This was somewhat unexpected, as it was felt that proportionately there would be a larger amount of quartz and mica in the oversized material.

Although this presumption is not supported by the flow-sheet analyses given, it is probable that other analyses will show some change, as mica and quartz are less easily reduced to small sizes than is calcite.

The great change in mineral composition takes place in the flotation cells. Here there is a fairly complete separation of the calcite and dolomite from all the other minerals. The reagents used tend to float the calcium carbonate minerals while all of the others sink to the bottom. Any carbonaceous matter is also floated, but in the Valley Forge Cement Company's rock there is too small an amount to receive consideration.

Photomicrographs of the Valley Forge Cement rock, the head sample of the flotation cells, the concentrates and rejects are shown. It must be borne in mind that the minerals in the original rock as well as the other samples are not uniformly distributed, so that no two specimens would present the same appearance. Those shown are fairly typical, although the view of the raw rock shows an excessive amount of impurities. This excess, however, is not as great as appears, because the mica occurs in very thin flakes.

EXPERIMENTAL PROCESSING OF LEHIGH VALLEY CEMENT ROCK

With the achieved success in processing the micaceous limestone of the Valley Forge Cement Co., it was natural to direct attention to similar treatment of the argillaceous limestone of the Lehigh Valley, so extensively utilized in the manufacture of Portland cement. The two limestones are similar in chemical composition, of the same age and formed in the great inland sea that covered most of Pennsylvania during the Ordovician period. In appearance they are so different that few persons might suspect a common origin. The dissimilarity in appearance is due to much greater compression and consequent greater heating to which the Chester Valley rocks were subjected than those of the Lehigh Valley. During this time of metamorphism some new minerals were formed but the principal change was to increase the size of the individual grains of all the minerals. Thus the sericite mica flakes of the Lehigh Valley cement rock are so small as to be separately invisible to the naked eye, while in the Chester Valley cement rock they are readily visible. The calcite, dolomite and quartz grains are also much larger in the more highly metamorphosed rock. Originally there was probably about the same amount of carbonaceous material present in both places but much of this was eliminated in the Chester Valley rocks, so that they are noticeably lighter in color than the argillaceous limestones of the Lehigh Valley.

Up to the present the work done in the Lehigh Valley rocks has been limited to experiments, no commercial installation having been made. The finer size of grains has called for some variations in the process as worked out for the Chester Valley rock, but no fundamental changes seem

necessary. All results obtained thus far indicate that it will be both feasible and economically advisable to install processing plants to treat this finer grained rock.

The method of experimentation was the same as that described for the Valley Forge rock. Three samples were tested (Table 6).

TABLE 6.—*Analyses of Lehigh Valley Rocks*

	Sample No. 1		Sample No. 2		Sample No. 3	
	Raw Rock	Concentrates	Raw Rock	Concentrates	Raw Rock	Concentrates
SiO ₂	14.28	7.20	16.40	9.66	18.76	5.50
Fe ₂ O ₃	1.13	1.17	1.70	1.34	1.67	1.27
Al ₂ O ₃	5.17	2.63	5.50	3.80	6.01	4.20
CaO.....	43.36	49.54	39.99	45.62	38.04	47.65
CaCO ₃	77.43	88.46	71.41	81.46	67.93	85.09
MgO.....	0.74	0.72	1.42	1.46	2.09	1.53
MgCO ₃	1.54	1.51	2.96	3.06	4.37	3.20
Loss.....	34.50	38.53	33.48	38.00	32.43	39.20

In every sample of Lehigh Valley cement rock examined the froth flotation process was successful. In some specimens the excess of carbonaceous material interfered with the flotation of the calcite. It was largely removed in a preliminary cell.

EXPERIMENTAL PROCESSING RESULTS ON A MIDDLE WESTERN SILICEOUS LIMESTONE

A limestone from the Middle Western portion of the United States was processed by the method described, with the results shown in Table 7.

TABLE 7.—*Processing of Middle Western Siliceous Limestone*

	Raw Stone	Concentrates
SiO ₂	27.66	6.10
Fe ₂ O ₃	0.29	0.92
Al ₂ O ₃	2.59	2.38
CaO.....	38.19	50.85
CaCO ₃	68.19	90.80
MgO.....	0.51	Not determined
MgCO ₃	1.06	Not determined
Loss.....	30.30	39.30

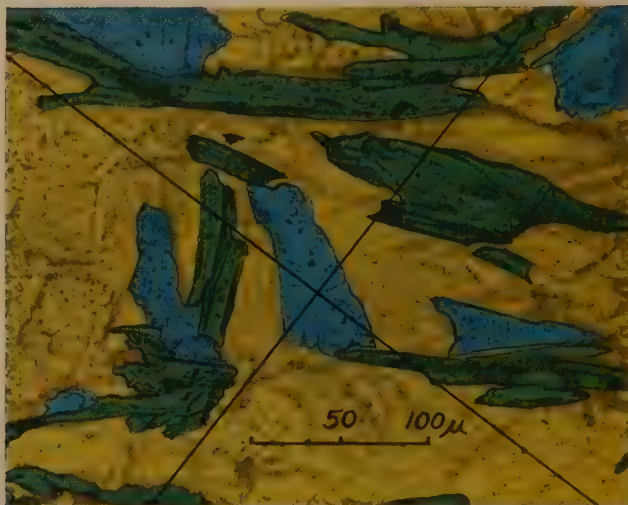


FIG. 5.—VALLEY FORGE CEMENT ROCK, A METAMORPHOSED MICACEOUS LIMESTONE OF APPROXIMATELY SAME GEOLOGICAL AGE AS LEHIGH VALLEY CEMENT ROCK.

Calcite (yellow) is medium to fine grained. Alumina is present as relatively large flakes of muscovite (green) between which are thicker layers of carbonate minerals, together with quartz grains (blue). Analysis of specimen is: SiO_2 , 15.56 per cent; Fe_2O_3 , 1.69; Al_2O_3 , 6.13; CaO , 39.53.

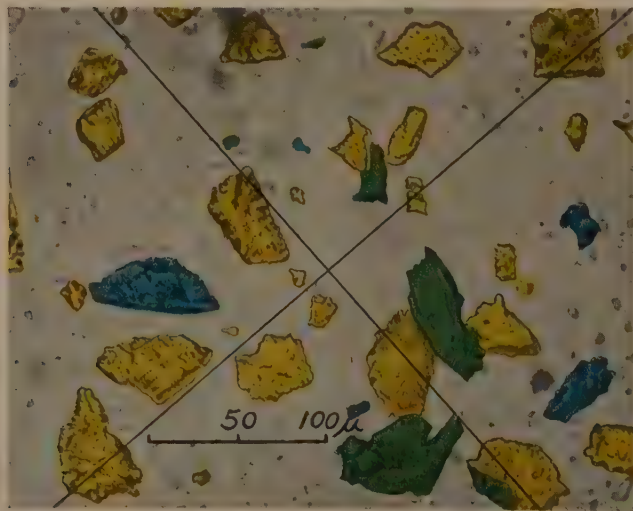


FIG. 6.—HEAD SAMPLE AFTER PARTICLE-SIZE CLASSIFICATION BUT BEFORE FLOTATION PROCESSING.

Yellow indicates calcium (and magnesium) carbonate; blue, quartz; green, mica. Some carbonaceous inclusions. Notice one grain of mica not separated from carbonate. One fairly large gray grain in lower right corner is iron oxide. Approximate analysis is: SiO_2 , 15.56 per cent; Fe_2O_3 , 1.69; Al_2O_3 , 6.13; CaO , 39.53.

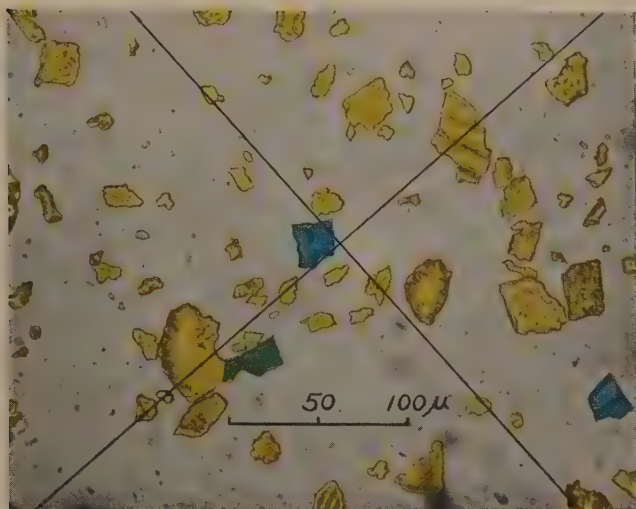


FIG. 7.—FLOTATION CONCENTRATES. PRODUCT CHIEFLY CALCITE.

Yellow indicates calcium (and magnesium) carbonate; blue, quartz; green, mica. A few small grains of quartz and mica were trapped and not eliminated. Approximate analysis is: SiO_2 , 3.82 per cent; Fe_2O_3 , 1.22; Al_2O_3 , 2.52; CaO , 47.98.

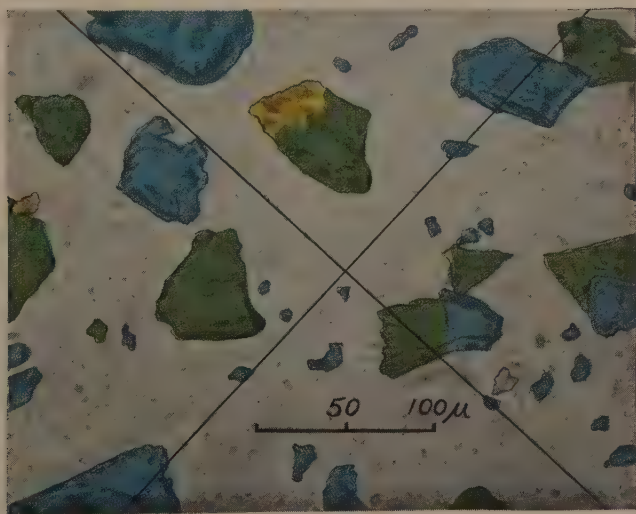


FIG. 8.—FLOTATION REJECTS, ALMOST ENTIRELY QUARTZ AND MICA.

Yellow indicates calcium (and magnesium) carbonate; blue, quartz; green, mica. Reasons why some calcite is lost with rejects will be apparent from the particle at the right center. Physical bond between carbonate and quartz was not broken by grinding. In treatment of this rock, however, grinding to abnormal fineness would cost much more than the value of the lost calcite. Approximate analysis is: SiO_2 , 57.76 per cent; Fe_2O_3 , 1.04; Al_2O_3 , 20.42; CaO , 6.23.

EXPERIMENTAL PROCESSING OF AN AUSTRALIAN MARL

A sample of a shell marl high in silica from Australia yielded the results given in Table 8.

TABLE 8.—*Percentages Obtained with an Australian Marl*

	Raw Marl	Concentrates	Rejects
SiO ₂	28.50	4.90	70.84
Fe ₂ O ₃	0.75	0.69	1.01
Al ₂ O ₃	5.65	2.54	10.65
CaO.....	34.33	50.40	6.04
CaCO ₃	61.30	90.00	10.78
MgO.....	1.02	1.84	0.31
MgCO ₃	2.13	3.85	0.65
Loss.....	28.45	39.50	7.25

SUMMARY OF EXPERIMENTS

The examples of limestones of various kinds given in this paper seem to warrant the belief that the process now in use at the Valley Forge Cement Company's plant is applicable to practically any type of limestone in which quartz, mica and iron oxides exist. Other noncalcareous minerals present in objectionable quantities probably can be removed with equal ease.

Limestone that is poorly adapted for cement because of low lime content or improper ratios of silica, alumina and iron may be corrected. Some of the products removed may be returned to the mix in the desired proportions or the requisite amounts of clay, iron ore or silica from other sources can be added. It seems possible, by proper processing, for a cement company to make from the same type of quarry rock not only the ordinary portland cement but practically all the special kinds of cements, with only minor amounts of foreign material, if any, for special correction. A number of formulas have been worked out to show how it is possible to make proper blends of concentrates and fine slurry for special cements, but these are omitted from this paper. Any cement chemist can readily devise combinations for the different grades of cement.

Because froth flotation for limestone processing started in the cement industry, the examples cited have all been taken from that branch of industry. It seems probable that most other types of limestone can be purified in the same general manner. Experiments are under way for the improvement of limestone for the chemical industry by the almost complete removal of all impurities. Results achieved are encouraging but not yet conclusive. If the process is successful it will be welcomed by the various companies that manufacture chemicals.

ADVANTAGES OF LIMESTONE PROCESSING

Several advantages of limestone processing have been suggested in the preceding discussion. These may now be summarized. Naturally, some of these apply to certain operations only.

1. *Quarrying Costs Reduced.*—In many limestone quarries there are poor ledges or spots where the stone, for some reason, is ill adapted for the purpose required. In some instances this poor stone is quarried and thrown away at considerable expense or the quarry is worked in an awkward and inefficient way in order to leave the poor stone untouched. Obviously, the utilization of this inferior stone after processing will effect quarry economies.

2. *Limestone Hitherto Unavailable Becomes Usable.*—In the efforts of all manufacturers to make their products in proximity to their markets, it seems possible that some limestone areas not now utilized may be developed.

3. *Costs of Grinding Raw Rock Reduced.*—In the grinding of limestone for cement it is recognized that the presence of coarse quartz and, to a lesser degree, mica and iron oxides materially increases grinding costs. The elimination of these by froth flotation facilitates grinding.

4. *Saving of Coal.*—If an excess of silica or coarse mica exists in the slurry, more heat in the kiln is necessary to produce the desired chemical combinations. Actual practice at the Valley Forge Company's plant shows saving of coal when using processed slurries.

5. *Costs of Clinker Grinding Reduced.*—Frequently coarse particles of quartz pass through the kiln unchanged. If this happens, the costs of clinker grinding is increased, whereas the removal of these from the slurry is advantageous.

6. *Available Supplies of Limestone Increased.*—Many lime and cement companies possess limited supplies of usable stone. The processing of poorer stone may greatly prolong the life of such operations.

7. *Elimination of High-grade Limestone Purchases.*—Many of the cement plants throughout the Lehigh Valley, as well as elsewhere, are compelled to purchase high-grade limestone from other sections to produce the proper mix. In some cases the amount purchased and shipped to the plant has run as high as 30 per cent. In some instances, this cost has become prohibitive. It seems probable that proper treatment of the quarry stone might have saved these plants by making unnecessary the purchase of stone from other sections.

8. *Improvement of the Quality of Cement.*—The processing of limestone has permitted the introduction of materials into the kiln in the form most readily combinable at the normal heat. The resulting cement shows less uncombined lime and silica and possesses greater ultimate strength as well as higher early strength.

9. *Enables a Company to Produce Different Products.*—The processing of limestone and later blending by the addition of the necessary materials permits a cement company to manufacture from the rock of a single quarry various types of cement as demanded.

10. *Sale of Materials Separated from the Limestone.*—Although it may seem visionary at the present time, the possibility of marketing some of the rejects is worthy of mention. Several uses for some of these products have been suggested.

DISADVANTAGES OF LIMESTONE PROCESSING

Naturally, there are some arguments that may be advanced in opposition to the installation and operation of a limestone processing plant. These are briefly summarized.

1. *Cost of Plant.*—The cost of a limestone processing plant must be considered because it requires an extra expenditure for equipment. So many factors enter into the costs, and these vary so widely because of specific requirements, that it is useless to offer any figures. It may be said, however, that the first costs are moderate.

2. *Cost of Operation.*—The costs of operation, involving labor, reagents, upkeep, interest on expenditure, depreciation, etc., will vary with almost every establishment, so that at present no figures can be cited. The Valley Forge Cement Co. has demonstrated the practicability of processing its stone, in that the decreased quarry, fuel, grinding and burning costs have more than offset the costs of operation of the plant. In addition, a higher grade of cement has been produced. With increased experience, the operation costs probably will be reduced substantially.

3. *Value of Stone Discarded.*—As shown on a preceding page, the processing thus far done has resulted in the discarding of some useful material. This is not great but should not be entirely overlooked. The Valley Forge Cement Co. has found that the tonnage of CaCO_3 wasted in the rejects is decidedly less than the low-grade quarry rock that previously was discarded.

CREDITS VERSUS DEBITS

The authors are convinced that the advantages far exceed the disadvantages, but it remains for each concern to evaluate properly both the credits and debits. Every limestone concern possesses its own problems, which demand individual examination and treatment. Each installation, to be most efficient, will require certain minor modifications of the practice described in this paper, but it is felt that these variations will be only slight and that the fundamentals described are sound and usable in practically every plant.

No mention has been made of the processing of limestone for use in a dry-process cement plant. This is under investigation but has not yet been perfected.

DISCUSSION

(*S. H. Dolbear presiding*)

J. C. WILLIAMS* AND G. M. DARBY,† Westport, Conn. (written discussion).—The application of flotation to the preparation of cement slurry at the plant of the Valley Forge Cement Co. is a striking example of how a method developed by one industry may be adapted with profit by another.

Ore-treatment plants adapted the tube mill from cement practice and in return gave classifiers, thickeners, agitators and filters developed in metallurgical work and at present also used in wet cement plants. Now flotation has been added to the list and will, undoubtedly, receive serious consideration for beneficiating cement rock.

The use of flotation as a means of improving or controlling the quality of non-metallic minerals is receiving increasing attention. Minerals that are now being treated by flotation in large tonnages include fluorspar, phosphate rock, apatite and calcite. Graphite and sulphur have also been floated commercially. With the exception of graphite and sulphur, it is believed that fluorspar was the first nonmetallic mineral to be floated commercially in the United States (or, with the exception of a small tonnage produced by the Consolidated Mining and Smelting Co., Trail, B. C., in the world). In the early part of 1929 the Franklin Fluorspar Co. (a subsidiary of the Aluminum Company of America) began production of acid-grade fluorspar at the rate of about 800 tons per month by a process developed by The Dorr Company, Inc.² Since that time flotation plants in Florida and Tennessee have been successfully treating increasing tonnages of phosphate rock and an apatite plant in Russia was reported in 1931 capable of treating 650 metric tons in each of two sections with five additional sections under construction.³ It is significant that basically all of these plants treat by flotation a feed from which a certain amount of the finest slime or colloidal material has been removed. Such desliming is generally accomplished in Dorr bowl classifiers or Dorr hydroseparators.

In the plant of the Valley Forge Cement Co. desliming results from treatment in a hydroseparator and a straight classifier acting as a unit. This combination provides a flexible means of varying the tonnage and character of feed sent to flotation, the amount of which is dependent upon the grade of untreated slurry (see Table 1).

The development of this process is an example of the cooperative manner in which research and testing should be carried out from the laboratory stage to plant operation. Mr. Breerwood conceived the idea that if it were possible mechanically to separate the individual constituent minerals (such as calcite, quartz, mica and pyrite) of the rock available at the Valley Forge quarry he could then recombine them in proper proportions to yield different types of cement of constant chemical and physical characteristics. To this end the services of the Westport laboratory of The Dorr Company, Inc. were engaged in August 1932 and classification tests were made on several barrels of cement slurry, with no appreciable changes in the chemical analyses of the various products.

* Flotation Engineer, The Dorr Company, Inc.

† Director, Westport Mill, The Dorr Company, Inc.

² U. S. Patent 1785992 (Dec. 23, 1930): J. C. Williams and O. W. Greeman.

³ N. K. Karchmer: Flotation of Apatite in Russia. *Eng. and Min. Jul.* (1932)

Flotation was then tried and was found to be eminently successful in the laboratory. The laboratory process was investigated from every angle, including the preparation of processed slurry that was burned to cement, the physical and chemical qualities of which were determined. These laboratory tests were made by batch methods and flotation tests were made on samples of $\frac{1}{2}$ to 1 kg. Samples of classified cement slurry were taken to the American Cyanamid Co. for its recommendations as to size and number of flotation cells.

The next step was to check the laboratory methods and results on a larger scale at the plant itself. A 4-ft. Dorr bowl and classifier was used to deslime several tons of slurry and the rake product was floated in batches in a 24-in. Fagergren flotation cell supplied by The American Cyanamid Co., which also furnished a member of its technical staff to assist in this and later stages of the investigation. These tests checked the laboratory tests closely and also developed some new points of interest.

With the above information, a small-scale continuous pilot plant was set up at the Valley Forge Cement Co. plant, West Conshohocken, Pa. The equipment included a 4-ft. Dorr bowl classifier, two 6-ft. and one 12-ft. thickeners, and ten 14-in. Fagergren flotation cells, together with auxiliary pumps, reagent feeders and other necessary adjuncts. With this setup a number of different flowsheets were tried out, the variation in nature and amount of reagents was studied, and other pertinent data secured. This plant had a capacity of about 12 to 24 tons of slurry (dry weight) per 24 hr. with 3 to 6 tons of feed to flotation. The plant was operated for six weeks. Two hundred tons (dry weight) of processed slurry were prepared, which was burned to cement. Details of the kiln operation were noted, as was the grindability of the clinker in the finishing (cement) mills. The resulting cement was subjected to complete chemical and physical testing.

Finally, the large plant, capable of treating 700 tons (dry weight) of slurry per 24 hr. was designed and put into operation, and it need only be said that results have bettered expectations.

The writers, having been associated with this problem from the laboratory through to the final plant, feel that this is a splendid example of cooperation on the part of The Valley Forge Cement Co., The American Cyanamid Co. and The Dorr Company, Inc. The officials of the Valley Forge Cement Co. are to be complimented for their courage and vision in carrying this process from the laboratory stages through the pilot plant and finally to the commercial plant.

S. H. DOLBEAR,* New York, N. Y.—I note that the finer limestone is untreated by the flotation process. How much by-passes the cells, and why?

C. H. BREERWOOD.—The siliceous impurities quartz and mica occur as relatively coarse and fine grains. The purpose is to remove the coarser grains, because they are most detrimental and increase the grinding and fuel costs. The overflow is fine enough to burn readily. Also, the relatively coarse material froth floats more readily than the fine.

The proportion by-passed depends on the composition of the quarry rock. The untreated rock averages about 70 per cent calcium carbonate, and it must be raised to 76. If the untreated rock runs higher than 70 per cent, the proportion by-passed may be increased, and, conversely, if it runs lower than 70 the amount by-passed must be decreased.

MEMBER.—Would gravity separation alone be successful for removing quartz and mica?

* Consulting Engineer.

J. C. WILLIAMS.—No, very little beneficiation could be thus attained.

W. F. POND,* Nashville, Tenn.—Do you now follow selective quarrying as you did before flotation was introduced?

C. H. BREERWOOD.—The process has enabled us to completely eliminate selective quarrying.

W. F. POND.—If the mill feed is variable, how do you correct it?

C. H. BREERWOOD.—After being ground the rock is pumped to storage and analyzed before processing. As the object in processing is to produce a composition (76 per cent CaCO_3) and knowing in advance the lime value of the rock to be processed, we correct the slurry by varying the quantity floated. For instance, if the rock to be processed analyzes 72 per cent CaCO_3 we float 40 per cent and mix the lime concentrate with the 60 per cent untreated to produce a perfect composition. If the rock is 68 per cent CaCO_3 we float 53.5 per cent, and so on, varying the quantity in strict accordance with the grade of the rock.

* State Geologist, Tennessee Division of Geology.

A Study of the Flotative Properties of Gypsum*

By W. E. KECK,† ASSOCIATE MEMBER A.I.M.E. AND PAUL JASBERG‡

(New York Meeting, February, 1937)

THERE is a considerable tonnage of iron ore in the Menominee Range of Michigan that is unsalable only because it has too large a content of sulphur. Beneficiation of such ore is economically desirable, as a relatively large quantity of finished concentrate would be obtained (deleterious sulphur mineral content being generally less than 5 per cent) and because large blocks of such ore have been developed incidentally during search for low-sulphur ore.

Research on the flotation beneficiation of these high-sulphur ores consisted of: (1) a study of the flotative properties of the principal minerals in a nearly pure state, (2) flotation of synthetic mixtures of these pure minerals and (3) application of this experimental information to samples of the ores.

Gypsum in iron ores is extremely deleterious; but when sufficiently free from impurities it is a valuable commodity. In 1934, the United States produced 1,500,000 tons of calcined and uncalcined gypsum valued at \$13,700,000¹. Of this quantity and value, Michigan produced approximately 19 per cent¹. Thus this investigation may also prove to be of interest in connection with the gypsum industry if minerals separation, such as flotation, becomes necessary in the future processing of gypsum. Data on the flotative properties of gypsum would also be of interest to the science of flotation because such information has been extremely limited. The technical literature reveals one reference to the flotation of gypsum, in which was discussed the use of the mineral, with several others, to determine the relation between the floatability of minerals and the pH of the flotation solution². This investigation necessarily was not comprehensive in regard to gypsum, but it did show that the mineral could be floated with oleic acid under varying pH conditions.

* This paper describes a part of a general investigation of the flotation of Michigan's potential iron ores. A comprehensive discussion of the status of these ores and of the reasons for research on their flotation beneficiation is available in mimeographed form on application to the Michigan College of Mining and Technology, Houghton, Michigan. Manuscript received at the office of the Institute Sept. 14, 1936. Issued as T.P. 762, MINING TECHNOLOGY, January, 1937.

† Research Engineer, Michigan College of Mining and Technology.

‡ Laboratory Research Assistant, Michigan College of Mining and Technology.

¹ References are at the end of the paper.

Gypsum is hydrated calcium sulphate ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$). The chemical composition and solubility (approximately 2 grams per liter of water solution³) indicate that gypsum is floated with difficulty. On the other hand, its flotation is favored by its relatively low specific gravity (approximately 2.3). The final and principal object of this work is the flotation separation of hematite and other economic iron minerals from gypsum. Thus the nonfloatability of gypsum is not a disadvantage if the iron minerals are readily floatable. Therefore both the flotation and the depression were studied.

PREPARATION OF THE PURE MINERAL

The ore used in this work was massive rock gypsum obtained from the United States Gypsum Co., Alabaster, Mich. It was composed of white and of dark colored material. The ore was crushed, and the dark lumps were rejected by hand picking.* The white lumps were further crushed and then ground (dry) to pass through 65 mesh. Comminution was carried out slowly and in stages to obtain as much +200-mesh material as possible and to prevent heating and dehydration of the mineral. The -200-mesh material was separated from the -65 + 200-mesh material by dry screening. The former size was rejected to prevent mechanical enmeshment of this fine material and the consequent introduction of error into the flotation results. Probability of mechanical enmeshment is relatively great with gypsum because of its low specific gravity.

Chemical analyses indicated that the -65 + 200-mesh gypsum contained 32.6 per cent lime, 47.5 per cent sulphur trioxide, 20.9 per cent chemically combined water, 0.04 per cent silica, traces of carbon dioxide, magnesia and iron oxide, and no chloride or native sulphur.† These quantities are nearly equal to the theoretical ones for gypsum; 32.5 per cent lime, 46.6 per cent sulphur trioxide, and 20.9 per cent chemically combined water.

Freshly broken hand specimens contained some crystals of anhydrite. Therefore a sink and float analysis was made with acetylene tetrabromide-acetone solutions. The specific gravities of gypsum and anhydrite are 2.3 and 2.9 respectively. The specific gravities of the solutions and the quantities of mineral that sank were as follows: 2.5 sp. gr., 0.01 per cent; 2.4 sp. gr., 0.03 per cent; 2.3 sp. gr., no separation and 2.2 sp. gr., 99.15 per cent. All of these data show that the purified gypsum contained only

* Preliminary flotation tests showed that the dark grains of gypsum floated readily and that the white ones did not. This fact indicated that the dark grains contained impurities that activated their flotation, therefore the dark material was rejected.

† Analyzed according to Standard Methods of Testing Gypsum and Gypsum Products [Amer. Soc. Test. Mat. Standards (1933) 96, II, Non-Metallic Materials] except that the sample was dried at room temperature to remove free water⁴.

small quantities of anhydrite. Because anhydrite changes to gypsum when in contact with atmospheric moisture, the surfaces of the anhydrite grains were probably covered with a film of gypsum, a condition that would not cause error in the results of flotation.

A pulp was made of 50 grams of the gypsum and 200 c.c. of distilled water. The filtrate from the pulp had a pH of 6.9. After the same 50 grams of mineral had been treated with three 200-c.c. portions of distilled water, the pH of the filtrate remained constant at 6.4 to 6.5. Incidentally, the first filtrate was also analyzed qualitatively for chloride and a negative test was obtained.

The data of all of these examinations indicated that the gypsum of -65 + 200-mesh was sufficiently pure for the proposed research, therefore it was used exclusively in this work.

FLOTATION PROCEDURE

All of the experiments were performed in a 50-gram enameled flotation cell, which has been described by M. S. Hansen⁵. The flotation reagents were chemically pure except the following: U.S.P. cresylic acid, oleic acid and lime; technical n-amyl alcohol, sodium silicate; practical gelatin; and sodium lauryl, oleyl and octadecyl sulphates of unknown purity. Reagents were dissolved in water unless otherwise indicated. Quantification of the reagent was obtained by the addition of measured volumes of the solutions. The xanthates, sodium salts of the fatty acids, and sodium salts of the sulphated alcohols were dissolved in boiled distilled water, and the solutions were kept in an atmosphere free of carbon dioxide. The reagents and ore were added in the following order: (1) distilled water; (2) activator, depressor or conditioner; (3) ore, collector, frother, and distilled water. The total volume of the liquid portion of the pulp was 85 c.c. during conditioning and approximately 165 c.c. during flotation. The pulp was conditioned for 5 min. and froth was collected for 5 min. The froth (water and floated mineral) was weighed. All gypsum products were dried at room temperature to avoid dehydration of the mineral. Experiments were conducted at room temperature, 20° to 22° C., unless otherwise indicated. The pH of the tailing solution was determined with a colorimetric comparator.

PRESENTATION AND DISCUSSION OF DATA

The quantities of reagents are expressed in pounds per ton of gypsum or, if comparison on a molecular or equivalent basis was desirable, in pound mols or pound equivalents per ton. The amount of gypsum floated is given in percentage of the total weight of the concentrate and tailings products.

The scope of this work did not permit comprehensive study of all the detailed phases of this problem that were discovered in this first

general investigation. The hypotheses submitted will perhaps be of value if further and more specialized research is undertaken.

The flotative properties of gypsum were studied with various flotation reagents, therefore this work is classified and discussed from the standpoint of reagents.

EXPERIMENTATION

Frothers Only.—The floatability of gypsum was determined with 0.10, 0.20, 0.30, 0.40 and 0.50 lb. of each of the following reagents: terpeneol, cresylic acid, n-amyl alcohol and n-heptyl alcohol. The floatability was also determined with 0.05, 0.10, 0.15, 0.20 and 0.25 lb. of n-nonyl alcohol. Instead of pine oil, the usual frother, terpeneol was used⁶. No collection was noted with these reagents, and negligible quantities of gypsum were carried over mechanically. The use of all these reagents except n-nonyl alcohol produced a satisfactory volume of froth. No frothing was obtained with n-nonyl alcohol when more than 0.10 lb. was used, and only a small quantity of unstable froth was produced with 0.05 lb. of the alcohol. This nonfrothing property may be attributed to the relative insolubility of n-nonyl alcohol.

Xanthates.—The floatability of gypsum with the xanthates was studied to determine whether gypsum could be floated with the higher xanthates or with combinations of the xanthates and metal salt activators. True flotation was not observed with any of the quantities of xanthates indicated in Table 1. A small amount of gypsum was always carried over mechanically with the froth, a phenomenon that was more pronounced with the higher xanthates because these reagents, functioning as frothers, increased the volume of froth and incidentally the quantity of gypsum carried over mechanically.

TABLE 1.—*Flotation of Gypsum with 0.10 Pound of Terpeneol and Xanthates*

XANTHATE	POUNDS XANTHATE PER TON GYPSUM	pH RANGE
K Ethyl	0.1, 0.2, 0.3, 0.4, 0.6, 0.8, 1.0	7.2-7.4
K n-amyl	0.1, 0.2, 0.3, 0.4, 0.6, 0.8, 1.0	7.1-7.5
K n-hexyl	0.05, 0.1, 0.2, 0.3, 0.4, 0.5, 0.6	7.1-7.3
K n-heptyl	0.05, 0.1, 0.2, 0.3, 0.4, 0.5, 0.6	7.2-7.4
K Octyl	0.05, 0.1, 0.2, 0.3, 0.4, 0.5, 0.6	7.1-7.4
K Nonyl	0.05, 0.1, 0.2, 0.3, 0.4, 0.5, 0.6	7.2-7.4
K Lauryl	0.05, 0.1, 0.2, 0.3, 0.4, 0.5, 0.6	7.1-7.4

Xanthates and Metal Salts.—The kinds and quantities of the xanthates and of the metal salts used in the xanthate-metal salt combinations are shown in Table 2. A small quantity of gypsum was floated with potassium lauryl xanthate and 0.05 lb. of cadmium sulphate. Larger quantities of cadmium sulphate entirely depressed the gypsum. Flotation was not obtained with any of the other combinations of xanthates and metal salts indicated in Table 2. Several of the salt-xanthate combinations produced insoluble xanthates. However, gypsum was not floated because the insoluble xanthate apparently was not adsorbed on the mineral.

Sodium Oleate and Potassium Palmitate.—The solubility of gypsum probably would seriously hamper selective flotation with sodium oleate or sodium palmitate, because the corresponding calcium soaps are relatively nonselective. Nevertheless, the flotative properties of gypsum were studied with these materials in order to cover the field of usual reagents as completely as possible. Further, it was thought that the use of other reagents in conjunction with sodium oleate or potassium palmitate would perhaps overcome the difficulty. As shown in Fig. 1, gypsum is readily floated with sodium oleate or potassium palmitate. Experimentation with sodium oleate was continued because this reagent floated more gypsum than did potassium palmitate.

TABLE 2.—*Flotation of Gypsum with 0.10 Pound of Terpeneol, 0.50 Pound of Xanthates, and Metal Salts*

XANTHATE	METAL SALT	POUNDS METAL SALT PER TON GYPSUM								PH RANGE	
K Ethyl	$\text{Cu}(\text{NO}_3)_2 \cdot 3\text{H}_2\text{O}$	0.06, 0.1,	0.2,	0.4,	0.6,	0.8,	1.2			7.3–5.9	
K Ethyl	$\text{Pb}(\text{NO}_3)_2$	0.08, 0.2,	0.3,	0.5,	0.7,	1.0,	1.7			7.3–6.7	
K n-amyl	$\text{Hg}(\text{NO}_3)_2$	0.08, 0.2,	0.3,	0.5,	0.7,	1.0,	1.7			7.2–4.9	
K n-amyl	$\text{HgNO}_3 \cdot 2\text{H}_2\text{O}$	0.05, 0.1,	0.2,	0.3,	0.4,	0.6,	1.0			7.3–6.4	
K n-amyl	$3\text{CdSO}_4 \cdot 8\text{H}_2\text{O}$	0.05, 0.1,	0.2,	0.3,	0.4,	0.5				7.2–7.1	
K n-amyl	$\text{Zn}(\text{NO}_3)_2 \cdot 6\text{H}_2\text{O}$	0.07, 0.15,	0.30,	0.45,	0.60,	0.90				7.3–7.1	
K n-hexyl	$3\text{CdSO}_4 \cdot 8\text{H}_2\text{O}$	0.05, 0.1,	0.2,	0.3,	0.4,	0.5,	1.0			7.3–7.1	
K n-hexyl	$\text{HgNO}_3 \cdot 2\text{H}_2\text{O}$	0.05, 0.1,	0.2,	0.3,	0.4,	0.5,	1.0			7.1–6.4	
K n-hexyl	$\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$	0.06, 0.1,	0.2,	0.4,	0.5,	0.7,	1.2			7.3–7.2	
K n-hexyl	$\text{Hg}(\text{NO}_3)_2$	0.08, 0.2,	0.3,	0.5,	0.7,	1.0,	1.7			7.1–4.7	
K n-heptyl	$\text{Cu}(\text{NO}_3)_2 \cdot 3\text{H}_2\text{O}$	0.06, 0.1,	0.2,	0.4,	0.6,	0.8,	1.2			7.2–6.5	
K n-heptyl	$\text{Pb}(\text{NO}_3)_2$	0.08, 0.2,	0.3,	0.5,	0.7,	1.0,	1.7			7.2–6.5	
K Octyl	$\text{Fe}_2(\text{SO}_4)_3 \cdot 9\text{H}_2\text{O}$	0.03, 0.07,	0.2,	0.3,	0.4,	0.5,	0.7			7.0–4.1	
K Octyl	$\text{Ni}(\text{NO}_3)_2 \cdot 6\text{H}_2\text{O}$	0.15, 0.3,	0.6,	0.9,	1.2,	1.3,	1.5			7.4–7.2	
K Octyl	AgNO_3	0.1,	0.3,	0.5,	0.7,	1.0,	1.2,	1.7		7.4–6.3	
K Nonyl	$3\text{CdSO}_4 \cdot 8\text{H}_2\text{O}$	0.05, 0.1,	0.2,	0.3,	0.5,	0.8,	1.0			7.4–7.1	
K Lauryl	$3\text{CdSO}_4 \cdot 8\text{H}_2\text{O}$	0.05, 0.1,	0.2,	0.3,	0.5,	0.8,	1.0			7.3–7.1	
K Lauryl	$\text{Al}(\text{NO}_3)_3 \cdot 9\text{H}_2\text{O}$	0.06, 0.1,	0.4,	0.6,	0.9,	1.1,	1.3			7.3–5.2	
K Lauryl	$\text{CoSO}_4 \cdot 7\text{H}_2\text{O}$	0.07, 0.1,	0.3,	0.4,	0.7,	1.0,	1.4			7.4–7.1	

Sodium Oleate and Usual Frothers.—Sodium oleate is both a collector and frother. However, a better quality of froth and a better selectivity are generally obtained if an optimum quantity of another frother, such as pine oil, is also used⁷. It is shown in Fig. 2 that small quantities of the frothers increased the floatability of gypsum, the increase in flotation being due to a better quality of froth. The froth made with sodium oleate only was so intensely flocculated and it was floated with difficulty, a condition that was remedied by an addition of terpeneol or cresylic acid. The difference in flotation with these two frothers probably resulted from the greater solubility of cresylic acid*—the solubility of cresylic acid was too great for best frothing.

*The solubilities of ortho, meta, and para cresols are 24.5, 21.8 and 19.4 grams respectively per liter of water at 20° C.⁸ The solubilities of alpha and beta terpeneols are 1.98 and 2.20 grams respectively per liter of water at 15–20° C.⁹

Sodium Oleate and Alcohols.—The results of this experimentation are shown in Fig. 3. The data obtained with terpineol have been included to show its frothing properties as compared with those of the alcohols.

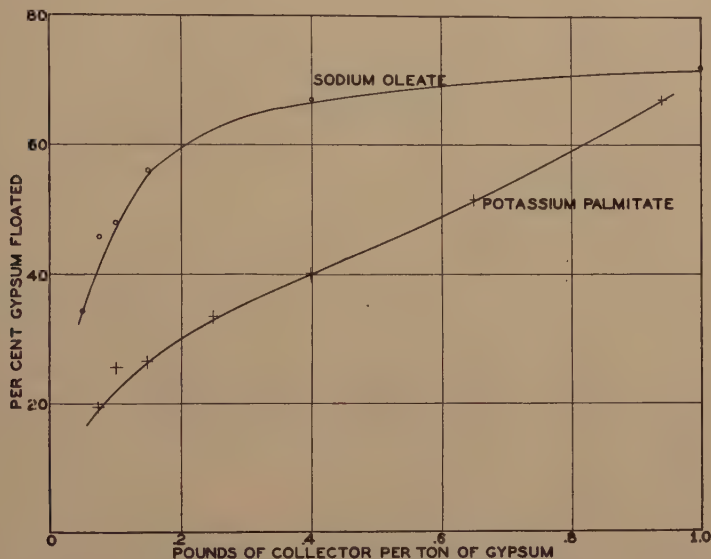


FIG. 1.—FLOTATION OF GYPSUM WITH SODIUM OLEATE AND POTASSIUM PALMITATE.

The experiments with methyl and ethyl alcohols were performed to determine whether their use as solvents for other reagents had any effect on flotation. When these alcohols were used in relatively large quantities flotation was affected (69 per cent of the gypsum was floated with 60 lb.

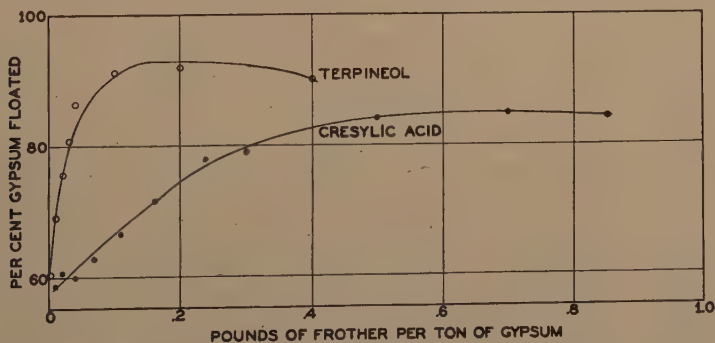


FIG. 2.—FLOTATION OF GYPSUM WITH 0.15 LB. SODIUM OLEATE AND TERPINEOL OR CRESYLIC ACID.

of methyl alcohol per ton of gypsum), therefore they were not used as solvents. With very small quantities of the alcohols, increase of flotation with the number of carbon atoms was perhaps due to the surface tension relations—reduction of surface tension and frothing increased with

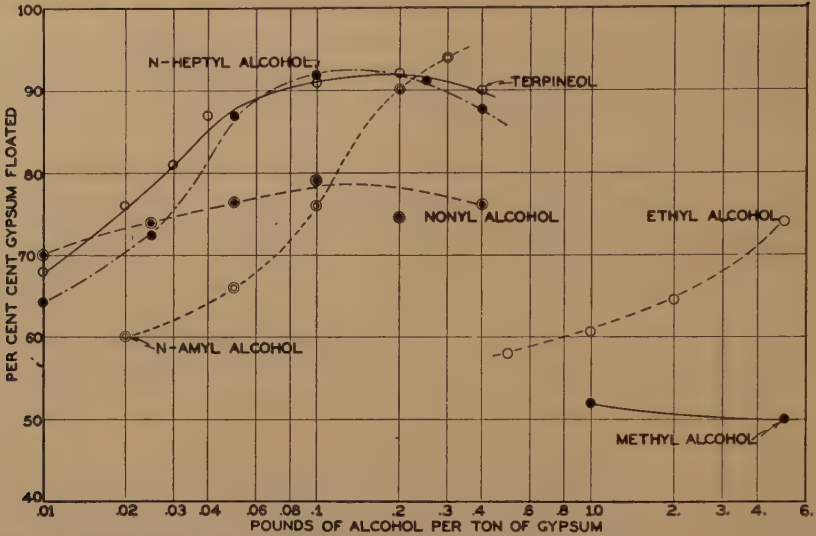


FIG. 3.—FLOTATION OF GYPSUM WITH 0.15 LB. SODIUM OLEATE AND ALCOHOLS.

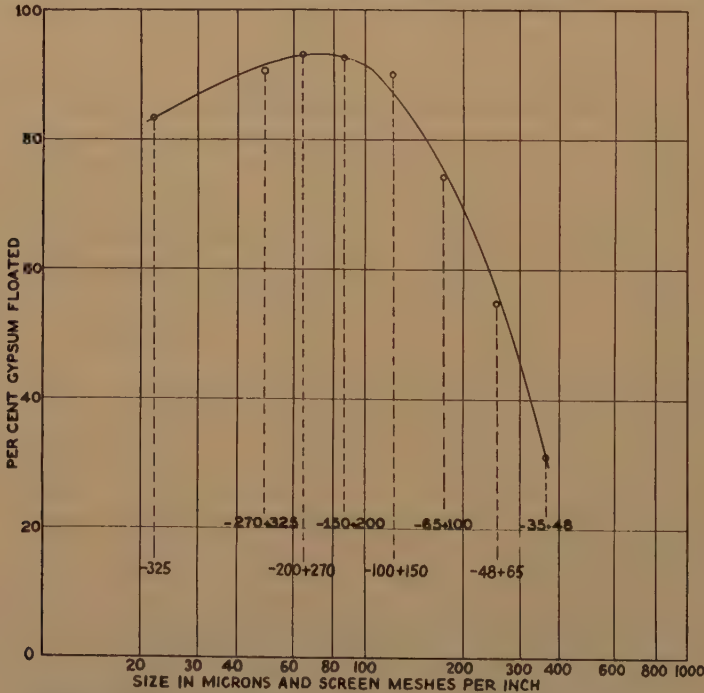


FIG. 4.—RELATION BETWEEN GRAIN SIZE AND FLOATABILITY.
Gypsum floated with 0.15 lb. sodium oleate and 0.03 lb. ter pineol.

the number of carbon atoms. When larger quantities were used, *n*-amyl and *n*-heptyl alcohols were the best frothers because the soluble and insoluble parts of their molecules were most properly balanced and because an optimum quantity was available for frothing. For good frothing, nonyl alcohol was too insoluble and methyl and ethyl alcohols were too soluble. The decreased flotation with the largest quantities of terpineol, nonyl alcohol, and *n*-heptyl alcohol was due to the increase in the thickness of the frother film over the optimum thickness.

Grain Size and Floatability.—Fig. 4 shows that $-35 + 48$ -mesh gypsum was least floatable and that maximum floatability occurred at

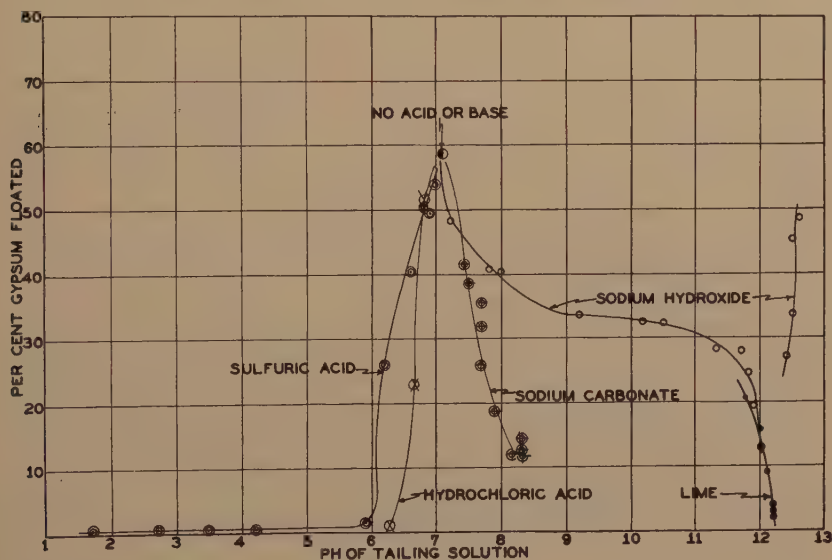


FIG. 5.—RELATION BETWEEN pH AND FLOATABILITY.

Gypsum floated with 0.10 lb. sodium oleate, 0.01 lb. terpineol and acidic or basic reagents.

150 to 270 mesh. These data indicate that if it is to be floated under these conditions the gypsum should be ground at least through 65 mesh.

Sodium Oleate and pH.—The data from this experimentation are shown in Figs. 5 and 6. Flotation was completely prevented by a small reduction in pH (to 5.9) because of the presence of either 0.4 lb. sulphuric acid or 0.5 lb. hydrochloric acid. Frothing was not seriously affected under these conditions. When the pH was increased by the addition of sodium hydroxide, flotation of the gypsum was first gradually depressed until approximately 10 lb. of the basic reagent was added, then it was activated with larger quantities of sodium hydroxide. With sodium carbonate, the pH of the tailing solution was increased only slightly, this chemical compound reacting with gypsum to form calcium carbonate and sodium sulfate. Frothing increased with the quantity of sodium carbon-

ate. A pH of 12.2 was obtained with 6.0 lb. of lime and this pH remained constant with increasing quantities of the reagent. The limited solubility of lime prevented increase of the pH beyond 12.2. The sodium oleate was probably consumed by the finely divided, undissolved calcium hydroxide (added as milk of lime) and consequently the flotation of the gypsum was depressed. With the smaller quantities of lime, it is probable that the increase of pH also caused depression, as it did when sodium hydroxide was used. Frothing increased with the quantity of lime. This tendency was especially marked after the maximum pH of 12.2 had been attained. Stabilization of the bubble by the finely divided floated particles of calcium hydroxide increased frothing.

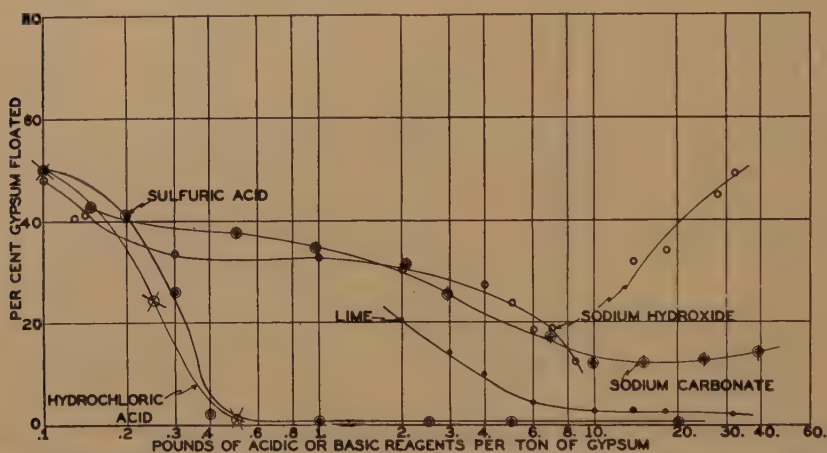


FIG. 6.—EFFECT OF ACIDIC AND BASIC REAGENTS ON FLOTATION OF GYPSUM WITH 0.10 LB. SODIUM OLEATE AND 0.01 LB. TERPINEOL.

Sodium Oleate and Metal Nitrates.—The results obtained in these experiments are shown in Figs. 7, 8 and 9. None of the metal salts were activators for gypsum, but several of them were depressors for the mineral. The data of Fig. 9 indicate that depression with the metal salts was caused largely by factors other than decrease of pH.

Lead nitrate had little effect on the flotation of gypsum. The following solubilities, based on one liter of saturated solution, are of interest in regard to lead nitrate: lead sulphate, 0.040 grams at 18° C. (ref. 8, 363); lead palmitate, 0.05 grams at 35° C. (ref. 8, 360); and calcium oleate, 0.065 to 0.091 gram, temperature not given (ref. 9, 1144). Lead oleate was assumed to be at least as soluble as lead palmitate. Comparison of these solubilities indicates that the lead was precipitated as the sulphate and that thus it probably did not enter directly into flotation. Similarly, magnesium oleate is more soluble than calcium oleate: magnesium oleate, 0.10 to 0.13 grams per liter of saturated solution at room temperature (ref. 9, 1284); calcium oleate, 0.065 to 0.91 grams per liter of

saturated solution, temperature not given (ref. 9, 1144). Insofar as the relative solubilities of calcium and magnesium oleate are concerned, magnesium also should have no direct effect on flotation. It is noted that other possible phenomena—for example, the formation of double

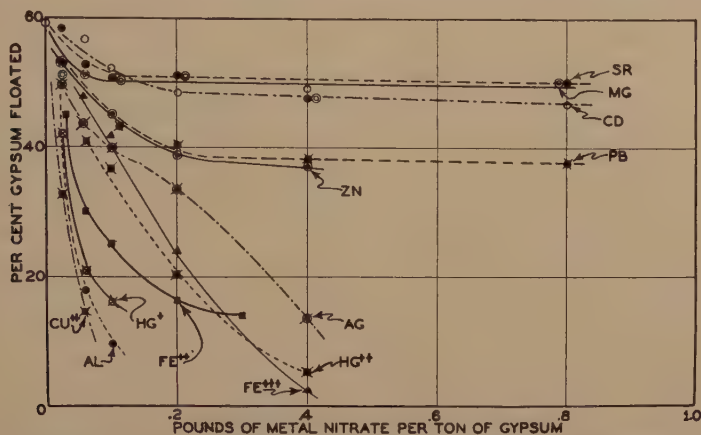


FIG. 7.—FLOTATION OF GYPSUM WITH METAL NITRATES, 0.10 LB. SODIUM OLEATE AND 0.01 LB. TERPINEOL.

salts of calcium and magnesium—would perhaps change the situation. Strontium, cadmium and zinc cannot be discussed at this time as were lead and magnesium because the necessary solubility data on the oleates of the metals are not available. Nor were such data obtained on the

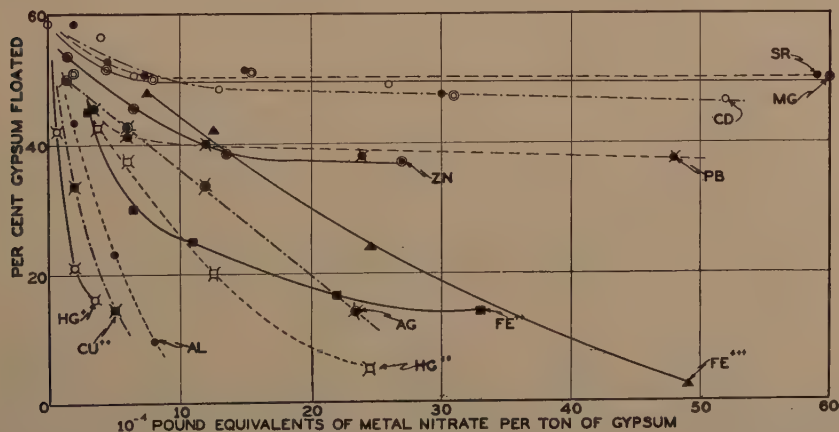


FIG. 8.—FLOTATION OF GYPSUM WITH METAL NITRATES, 0.10 LB. SODIUM OLEATE AND 0.01 LB. TERPINEOL.

oleates of the metals that markedly depressed gypsum. Thus it is difficult to classify these metal nitrates in regard to their effect on the flotation of gypsum. On the other hand, the depression observed with certain of the metal nitrates can be tentatively attributed to the non-

adherence of the particular metal soap on the gypsum. Decrease of pH may also, in certain cases, have caused some depression of the gypsum.

Sodium Oleate and Colloidal Reagents.—Frothing was little decreased by these colloidal reagents. Sodium silicate* was not an effective

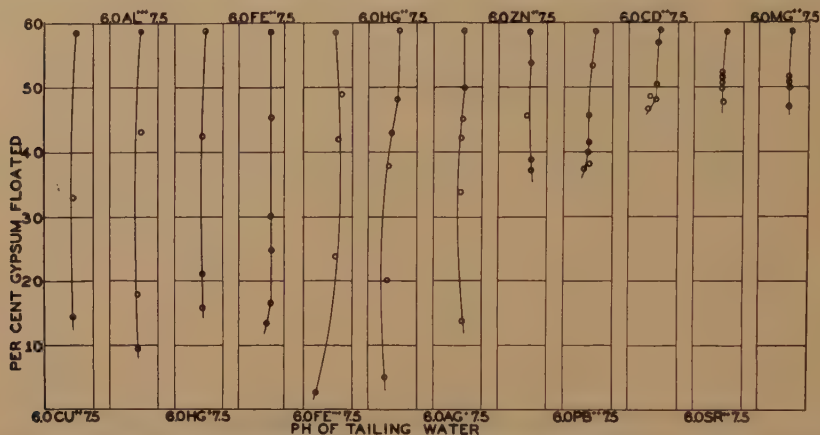


FIG. 9.—PH OF TAILING WATER. GYPSUM FLOATED WITH 0.10 LB. SODIUM OLEATE AND 0.01 LB. TERPINEOL AND METAL NITRATES.

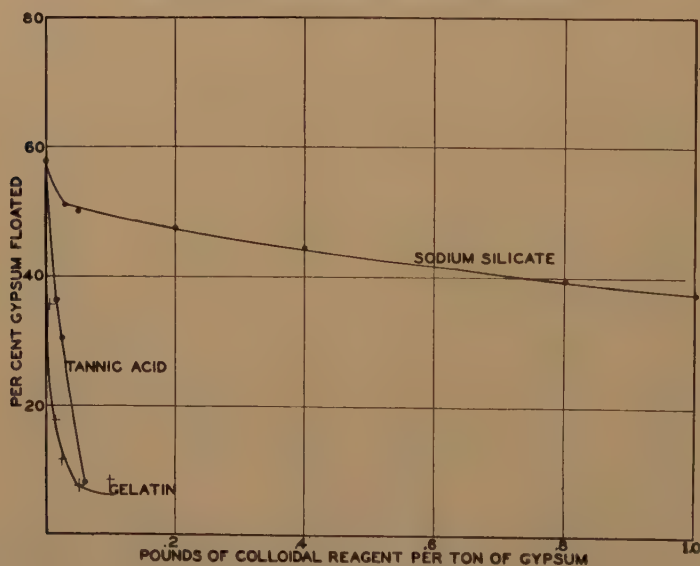


FIG. 10.—FLOTATION OF GYPSUM WITH 0.10 LB. SODIUM OLEATE, 0.01 LB. TERPINEOL AND COLLOIDAL REAGENTS.

depressor for gypsum (Fig. 10). On the other hand, gelatin and tannic acid were extremely potent depressors. The depression of flotation

* A technical brand of sodium silicate was used which is applied in flotation. The pH of the tailing solution varied from 7.0, when no sodium silicate was added, to 8.5 when 1.4 lb. of the reagent was used.

obtained with sodium silicate was apparently produced by an increase of pH. (Compare flotation at a pH of 8.5, which was due to sodium silicate, with flotation at a similar pH due to sodium hydroxide.)

Fatty Acids.—Oleic, palmitic and myristic acids were used in this experimentation. Because these fatty acids are relatively insoluble in it, water cannot be used as a solvent. Ethyl alcohol also was unsatisfactory for this purpose, because it increases flotation when used in the quantities required for a solvent. (1 c.c. of ethyl alcohol corresponded to approximately 32 lb. per ton.) The fatty acids are more soluble in ethyl ether than in ethyl alcohol; therefore less of the solvent can be used. Ether also volatilizes rapidly. For these reasons it was thought that flotation would not be much affected by the use of ethyl ether as a solvent.

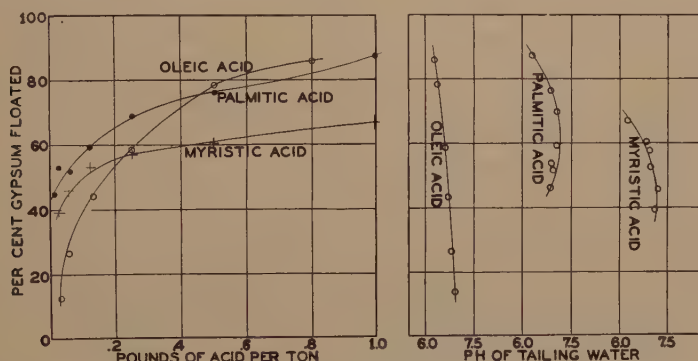


FIG. 11.—FLOTATION OF GYPSUM AND PH OF TAILING WATER WITH 0.02 LB. TERPINEOL AND FATTY ACIDS.

However, experiments showed that under the usual procedure ethyl ether also affected flotation to a considerable extent and was therefore not applicable.

Finally, the oleic acid was added in a constant quantity (0.10 c.c.) of ethyl ether solution. The data shown in Fig. 11 were obtained by introducing the fatty acids in this way. The quantity of acid was varied by preparing a different solution with the necessary acid concentration for each experiment, therefore the error due to the solvent was made constant and was kept relatively small.

Palmitic and myristic acids are solid at room temperature. Poor dispersion and little flotation are obtained with these reagents at ordinary temperatures, therefore flotation was conducted at temperatures from 7° to 8° above their respective melt points. The temperatures of flotation were as follows: oleic acid, room temperature; myristic acid, 61.5° to 62.5° C.; and palmitic acid, 68° to 69° C.

The solubility of gypsum changes only slightly within this range of temperatures. Some of these temperatures (Centigrade) and corresponding solubilities in parts per million are as follows: 20°, 2000; 40°, 2100;

60°, 2000 and 70°, 1900³. Further, gypsum is not converted to anhydrite between room temperature and 70° C.³ Therefore it is thought that the flotation data given above are quite accurately comparable.

When small quantities of these fatty acids were used, more flotation was obtained with the palmitic and myristic acids than with oleic acid. These results can be explained by assuming that the saturated palmitic and myristic acids are more insoluble than is the unsaturated oleic acid. Thus more of the palmitic and myristic acids are available for flotation after the solubility relations have been satisfied. The larger quantities of oleic and palmitic acids produced more flotation than myristic acid, probably because the hydrocarbon chain of the molecule of myristic acid was not long enough for best flotation.

Oleic acid floated less gypsum than did sodium oleate. The inferior results with oleic acid as compared to sodium oleate were probably due to the unavoidable acidic condition of the pulp, which tended to depress flotation. This tendency was shown in the experiments with sodium oleate and sulphuric or hydrochloric acids.

The difference in shape of the pH curves (Fig. 11) is caused by the fact that unboiled water was used in the oleic acid experiments, whereas the palmitic and myristic acid experiments were carried out with boiled water.* In the two series of experiments with palmitic and myristic acids carbonic acid had been at least partly removed; therefore, greater quantities of palmitic and myristic acids than of oleic acid were necessary to reduce the pH to the same extent.

Of interest in connection with the present consideration is the fact that X-ray study indicated the formation of calcium palmitate when palmitic acid was placed on gypsum¹⁰.

Sodium Lauryl, Octadecyl and Oleyl Sulphates.—Sodium salts of sulphated straight chain alcohols have been suggested as hard-water soap substitutes¹¹. The calcium salts of these soap substitutes are water-soluble. By the use of a reagent of this nature the disadvantage of the relatively nonselective calcium soaps of the fatty acids is avoided.

Data obtained from experimentation with three of these compounds† are presented in Fig. 12. The water solutions of these reagents were introduced into the flotation cell at temperatures from 45° to 50° C. The use of relatively large quantities of sodium lauryl and sodium oleyl sulphate effected quite complete flotation and also produced violent frothing. On the other hand, little flotation and a small quantity of

* Water at the boiling point and a bath were used to obtain the flotation temperatures previously noted for palmitic and myristic acids.

† The authors are indebted to R. A. Duncan, Proctor & Gamble Co., Ivorydale, Ohio, for samples and descriptions of these reagents. Sodium lauryl sulphate is the derivative of the saturated alcohols, C₁₀-C₁₄; sodium octadecyl sulphate is the derivative of the saturated alcohols, C₁₆-C₁₈; and sodium oleyl sulphate is the derivative of the unsaturated alcohols C₁₆-C₁₈.

intensely flocculated and heavily laden froth were obtained with sodium octadecyl sulphate.

First consideration of the hard-water soap substitutes suggested the conclusion that they are not collectors for gypsum, for the reason that

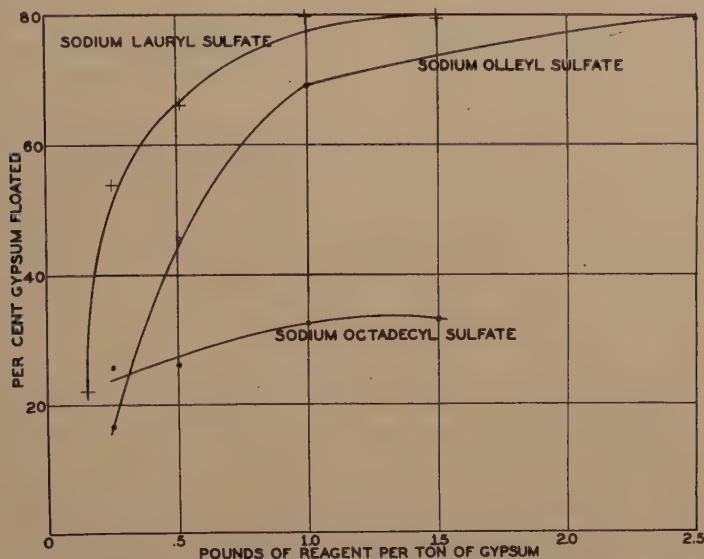


FIG. 12.—FLOTATION OF GYPSUM WITH SODIUM LAURYL, OCTADECYL AND OLEYL SULPHATES.

their calcium salts are too soluble. This tendency to depress the gypsum is overcome by unknown factors, as is shown by the data above. Nevertheless these reagents were further tested to determine whether they could be used to float gypsum and depress hematite.

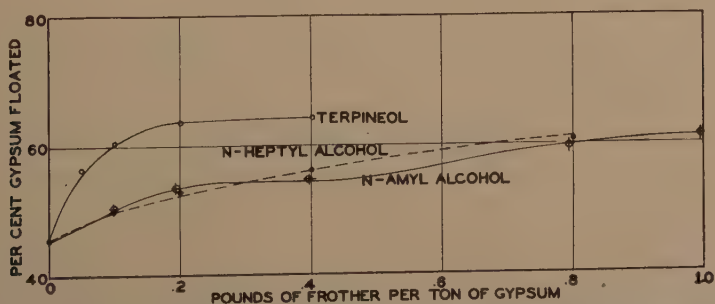


FIG. 13.—FLOTATION OF GYPSUM WITH 0.5 LB. SODIUM OLEYL SULPHATE AND FROTHERS.

Sodium Oleyl Sulphate and Frothers.—The data from this work are shown in Fig. 13. The use of sodium oleyl sulphate produced voluminous frothing; nevertheless the addition of other frothers increased flotation. Terpineol, amyl alcohol and heptyl alcohol were tested because their use with sodium oleate gave the greatest increase in flotation. Terpineol

was superior to the alcohols in this case just as it was when these frothers were used in conjunction with sodium oleate. One-tenth pound of terpineol was chosen as the optimum quantity and was used throughout the experiments with sodium oleyl sulphate.

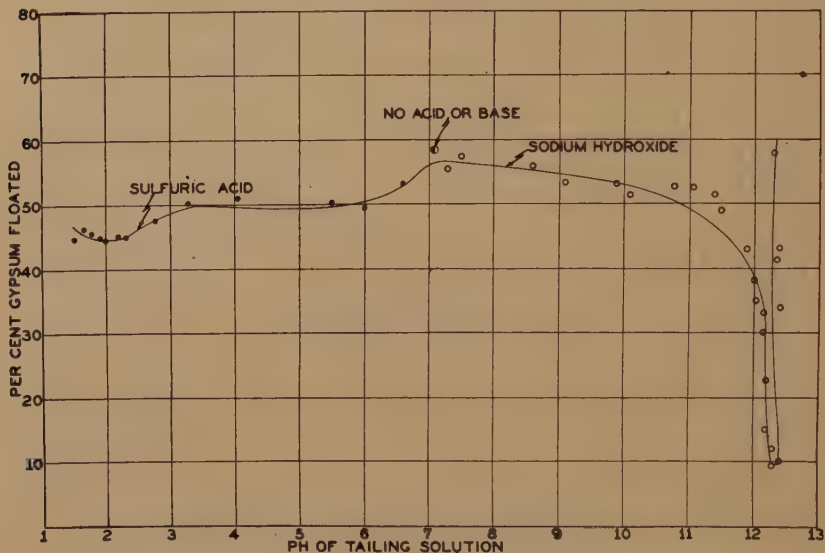


FIG. 14.—RELATION BETWEEN PH AND FLOATABILITY.

Gypsum floated with 0.5 lb. sodium oleyl sulphate, 0.1 lb. terpineol and acidic or basic reagents.

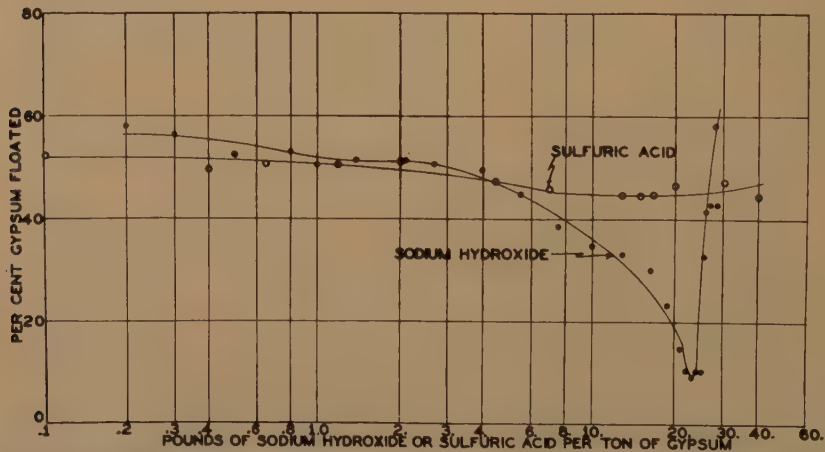


FIG. 15.—EFFECT OF ACIDIC AND BASIC REAGENTS ON FLOTATION OF GYPSUM WITH 0.5 LB. OF SODIUM OLEYL SULPHATE AND 0.1 LB. TERPINEOL.

Sodium Oleyl Sulphate and pH.—The desired acidity or alkalinity was obtained with sulphuric acid or sodium hydroxide. The data are shown in terms of pH in Fig. 14. Under very basic conditions, large changes in

flotation were obtained with little change in pH; therefore the data are also given in Fig. 15 in terms of pounds of acid or base. Flotation and frothing were little affected by acidic conditions induced by relatively large quantities of sulphuric acid. These results appear logical so far as stability of the sodium oleyl sulphate is concerned, since this reagent is stable under acidic conditions¹¹. Under basic conditions, the results obtained with sodium oleyl sulphate were similar to those obtained with sodium oleate.

The experiments with the hard-water soap substitutes concluded this general study of the flotative properties of gypsum. Data from this experimentation have been applied to the flotation of synthetic mixtures of gypsum and hematite and to samples of high-sulphur iron ores. This work will be published in the near future.

SUMMARY AND CONCLUSIONS

The flotative properties of gypsum were studied because this mineral is a source of sulphur in the high-sulphur iron ores of Michigan. A relatively pure natural gypsum was used in this investigation.

Gypsum was nonfloatable with xanthates prepared from monohydric saturated alcohols that have 2 to 12 carbon atoms. The mineral was also nonfloatable when combinations of these xanthates with several of the usual metal salts were used.

Excellent flotation was obtained with sodium oleate and palmitate. Terpeneol, *n*-amyl and *n*-heptyl alcohol when used in conjunction with sodium oleate materially increased the quantity of gypsum floated.

Small quantities of sulphuric or hydrochloric acid completely depressed flotation with sodium oleate. The use of increasing quantities of sodium hydroxide with sodium oleate first depressed and finally activated flotation. Both sodium carbonate and lime depressed flotation.

Several metal salts when used with sodium oleate strongly depressed the gypsum. Other salts had little effect on the flotation of the mineral and none activated it.

Sodium silicate with sodium oleate slightly reduced the quantity of gypsum floated. On the other hand, gelatin and tannic acid were extremely potent depressors.

Oleic, palmitic and myristic acids were not especially good collectors for gypsum, probably because an acidic condition was unavoidably produced.

The floatability of gypsum was tested with several hard-water soap substitutes.

The use of sulphuric acid had little effect on flotation with sodium oleyl sulphate. Sodium hydroxide produced the same results with sodium oleyl sulphate as it did with sodium oleate.

ACKNOWLEDGMENTS

The authors gratefully acknowledge the aid received in this work. They are especially indebted to G. C. Dillman, President, to N. H. Manderfield and C. T. Eddy, Associate Professors of Metallurgy, C. M. Carson, Professor of Chemistry and R. F. Makens, Instructor in Chemistry, of the Michigan College of Mining and Technology. The services of a laboratory assistant were provided under Project 31-F2-59 as a part of a cooperative research program sponsored by the Federal Emergency Relief Administration.

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Flotation of California Magnesites

By ERIC SINKINSON* AND S. D. MICHAELSON,† JUNIOR MEMBER A.I.M.E.

(New York Meeting, February, 1935)

MANY of the magnesite ores of the western part of the United States contain such large amounts of silica and hydrous silicate minerals that the value of the ores is either low or nominal. Expensive and carefully controlled mining methods are necessary to prevent excessive dilution of the higher grade ores by introduction in mining of small pieces of the hanging walls or footwalls of the deposit. This difficulty is further complicated by the various states of decomposition of the country rock of the deposits in which many of the ores occur.

The problem the authors attempted to solve arose from a request to float an impure magnesite containing varying proportions of siliceous gangue with calcium carbonate. The ore is normally enclosed in serpentine masses, somewhat decomposed, and in mining it wall rock is incidentally added, so that variable amounts of siliceous material occur in the samples. The silica, alumina, and iron oxide are so closely associated with the magnesium carbonate in the magnesite that ordinary mechanical separation at coarse mesh sizes is of no avail. It is well known that calcium carbonate can be floated by suitable reagents, therefore it seemed reasonable to suppose that magnesium carbonate minerals would behave in a similar manner, but experimental trial proved that this supposition was unfounded. Therefore the first step in the problem was to find out what differences of behavior there were between magnesium and calcium carbonates under flotation conditions, or when suspended in water.

In water at 18° C. magnesium carbonate is slightly more soluble than calcium carbonate; in fact, MgCO_3 is soluble to the extent of 0.0106 grams in 100 c.c. of water at 18° C. and CaCO_3 to the extent of 0.0013 grams. However, magnesium carbonate in water gradually changes to the basic carbonates of a composition $(3\text{MgCO}_3.\text{Mg}(\text{OH})_2.3\text{H}_2\text{O})$ or $(4\text{MgCO}_3.\text{Mg}(\text{OH})_2.5\text{H}_2\text{O})$, depending upon conditions, and calcium carbonate to calcium hydroxide, $\text{Ca}(\text{OH})_2$. The relative solubilities of these hydroxides in water are the reverse of the solubilities of magnesium and calcium carbonates. The range in published figures for solubility

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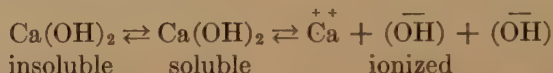
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of the basic magnesium carbonates is from 0.016 to 0.04 grams per 100 c.c. of water at 18° C., whereas the solubility of calcium hydroxide is 0.170 grams per 100 c.c. at the same temperature; sufficient, indeed, to impart actual alkalinity to the water.

It is understood that these reactions take place only at the surfaces of the respective mineral particles, which cause them to behave in a flotation cell as though the whole of the particles were of uniform composition. As the chemical reactions between mineral and water are progressive, they need control.

The course of the reactions of the hydroxides in water is



therefore an accumulation of hydroxide ions will increase the alkalinity of the water unless some neutralizing agent is present to prevent it. Any weak acid will furnish enough (H)⁺ ions to reduce this alkalinity. Thus it was decided to try the effect of carbon dioxide in the water for this purpose. The carbonic acid formed should tend to arrest the ionization of the calcium hydroxide and drive the reaction back to yield carbonate. As carbonic acid accumulates, from introduction of more CO₂, its further action should be to dissolve the normal calcium carbonate and form acid carbonate CaHCO₃, which has an increased solubility in water, but carbonic acid would react with the magnesium hydroxide to form the basic carbonate. To be considered also are the chemical effects of oleic acid, the collector, on the respective magnesium and calcium carbonates. There would no doubt be an inclination to form the insoluble calcium oleate on the calcium salt.

DESCRIPTION OF TESTS

The material actually tested showed on analysis a composition of: CaCO₃, 2.00 to 2.50 per cent; MgCO₃, 81.35 to 75.90; R₂O₃, 3.05 to 4.00; SiO₂, 13.60 to 17.60. All the ore samples were ground in an iron mortar.

Test 1.—A sample of the ore was ground dry to minus 100-mesh, the minus 200-mesh was screened out and the sample made up to a density of more than 10 per cent with distilled water saturated with carbon dioxide. This was allowed to stand for 16 hr. at room temperature. At the end of this time the pulp was diluted to a specific density of 10 per cent with water charged with carbon dioxide and was placed in a new Fagergren flotation unit. This unit, of 2½ liters capacity, was operated with the impeller rotating at approximately 2000 r.p.m. for a period of 3 min. and at a temperature of 20° C. The pH was 6.9. For a collector, oleic acid equivalent to 2.50 lb. per ton of dry magnesite was

added to the cell and the mass was agitated by the impeller with aeration by carbon dioxide for 3 min. The floated material resulting was analyzed and found to contain 91.8 per cent of total magnesium and calcium carbonates. As the feed contained only 87.5 per cent of the total magnesium and calcium carbonates, there was a concentration of 4.3 per cent over the feed. Previous attempts had been made to concentrate the magnesite with the same operating conditions except that no carbon dioxide was used, but with negative results. It appears therefore that carbon dioxide added to the water of the pulp before concentration definitely resulted in its beneficiation.

Test 2.—It was next decided to repeat test 1 but to lessen the time of exposure of the pulp to carbon dioxide from 16 hr. to 3 min. A fresh sample of the magnesite containing total carbonates of magnesium and calcium of 82.4 per cent was ground dry, the same sieve sizes as in the previous test being used. The sample was placed in water saturated with carbon dioxide for a period of 3 min. and made up to a specific density of 10 per cent, as before. The pH value was again 6.9. The flotation was conducted precisely as described in test 1, using CO_2 for aeration. The concentrate of total carbonates of magnesium and calcium were found to be 89.0 per cent, which is a concentration of 6.6 per cent over the feed. Moreover, an analysis of the calcium carbonate in the feed gave 2.47 per cent and the resulting concentration 1.16 per cent, a reduction of calcium carbonate in the concentrate of 1.31 per cent. Also, analysis showed that the silica and hydrous silicates present in the feed to the extent of 17.6 per cent had at the same time been reduced in the concentrate to 11.0 per cent, a reduction of 6.6 per cent.

It is evident that a brief exposure of the magnesite to carbonic acid is more beneficial than a prolonged one. This particular test bears out the suggestion made earlier in this paper that carbonic acid tends to render the magnesium carbonate basic and less soluble, hence more floatable, than the calcium carbonate, which, evidently owing to incipient formation of the bicarbonate salt, becomes relatively more soluble and less floatable than is possible without the application of carbon dioxide. The oleic acid used as collector appears not to interfere adversely in the test.

Test 3.—Another sample was prepared as before and made up to a specific density of 23 per cent with distilled water. There was no pretreatment of the pulp with carbon dioxide. The pulp was placed in the flotation cell with oleic acid to an amount equivalent to 1 lb. per ton of dry feed. The impeller was worked at 2000 r.p.m. for one minute while carbon dioxide was used as before for aeration of the cell. The results showed a concentration of total carbonates of magnesium and calcium of 5.1 per cent. The silica and hydrous silicates present in the feed were reduced in the concentrate by 5.1 per cent.

The alteration in the conditions of this test from those of test 2 resulted in lower yield of concentrates, as can be seen in Table 1.

Test 4.—Having thus far accomplished a successful concentration of the magnesium carbonate in the magnesite feed, it was decided to attempt refinements of procedure that might lead to better results. Sodium silicate was chosen because of its unusual properties in dispersing slimes and producing high-grade concentrates.

The sample was ground dry, as in the previous tests, made to a specific density of 18 per cent with water, placed in the cell with the addition of 1.00 lb. of oleic acid, 0.50 lb. of cresylic acid and 0.85 lb. of sodium silicate per ton of dry feed. The impeller was worked for 2 min. with CO₂ aeration, as described before. The total carbonates of magnesium and calcium in the feed were 85 per cent and the concentrate yielded 96.5 per cent total carbonates of magnesium and calcium, a concentration of 11.5 per cent of the total magnesium and calcium carbonates. The feed contained 2.48 per cent of calcium carbonate and the concentrate 1.13, a reduction of 1.35 per cent of calcium carbonate. The silica and hydrous silicates amounted to 15.0 per cent in the feed and only 3.5 per cent in the concentrate, a reduction of the silicates of 11.5 per cent.

Test 5.—This sample was prepared as in test 4 and the conditions of experiment were the same as described for that test, except that the quantity of sodium silicate was increased to the equivalent of 3 lb. per ton of dry feed. There resulted no concentration whatever. In fact, the grade of concentrate was lowered, as shown in Table 1. Undoubtedly, the excess of sodium silicate employed depressed the carbonates, one of the functions of sodium silicate.

Test 6.—In tests 1 to 5 the magnesite samples were ground dry. Later it was decided to grind the samples in water because the sample pretreated with water charged with carbon dioxide for only 3 min. gave a better all-round concentration than the sample pretreated for 16 hr. (Table 1), and it was thought that perhaps only the slightest exposure of the magnesite pulp to carbon dioxide incidental to the air and water would condition the surfaces of the magnesite particles during the grinding operation. To test this suggestion, the magnesite was ground down wet to 97 per cent through 100 mesh, made up to a specific density of 23 per cent with water and treated as the dry ground ore was treated, but at 23° C. Air was turned into the cell during agitation for 3 min. Oleic acid equivalent to 0.7 lb. per ton of magnesite was added. The feed to the cell contained 84.25 per cent of total magnesium and calcium carbonates; the concentrate, 94 per cent; a concentration of total carbonates of 9.8 per cent. There was 2.47 per cent of calcium carbonate in the feed and 1.16 per cent in the concentrate, giving a reduction in concentration of 1.31 per cent.

TABLE 1.—*Flotation Test Data*

Test No.	Temperature, Deg. C.	Pulp Density, Per Cent	pH	Grinding of Feed	Aeration in Cell	Reagents, lb. per Ton of Feed				Conditioning of Feed	Treatment Time, Min.	Total Carbonates, Per Cent		Increase of Carbonates in Concentrates over That in Feed	Increase of Carbonates in Concentrates over That in Original Amounts, Per Cent	Insoluble in Concentrates, Per Cent	Recovery of Total Carbonates, Per Cent
						Oleic Acid	Sodium Silicate	Cresylic Acid	Other Reagents			Feed	Concentrates				
1	20	10	6.9	Dry	CO ₂	2.50				16 hr. in H ₂ CO ₃	3	87.5	91.8	4.3	4.3 = 4.91 87.5	8.2	
2	20	10	6.9	Dry	CO ₂	2.50				3 min. in H ₂ CO ₃	3	82.4	89.0	6.6	6.6 = 8.2 82.4	11.0	72
3	20	23	7.2	Dry	CO ₂	1.00		1.50		3 min. in H ₂ CO ₃	1	85.0	90.1	5.1	5.1 = 6.0 85	9.9	77
4	20	18	7.2	Dry	CO ₂	1.00	0.85	0.50		3 min. in H ₂ CO ₃	2	85.0	96.5	11.5	11.5 = 13.5 85	3.5	83
5	20	18	7.0	Dry	CO ₂	1.00	3.00	0.50		3 min. in H ₂ CO ₃	2	85.0	84.5	-0.5	-0.5 = -0.588 85	15.5	69
6	23	25	7.2	Wet	Air	0.70				3 min. in H ₂ CO ₃	3	84.25	94.1	9.85	9.85 = 11.7 84.25	5.9	76
7	23	25	7.1	Wet	Air	0.70	0.50	0.35		3 min. in H ₂ CO ₃	3	84.25	92.0	7.75	7.75 = 9.2 84.25	8.0	84
8	23	25	7.2	Wet	Air	0.70	0.50	0.35	0.7 NaHCO ₃	3 min. in H ₂ CO ₃	3	84.25	90.1	5.85	5.85 = 6.95 84.25	9.8	80
9	30	12	7.2	Wet	Air	1.50				3 min. in H ₂ CO ₃	3	86.4	96.8	10.4	10.4 = 12.05 86.4	3.2	81
10	30	12	7.2	Wet	Air	1.50				3 min. in H ₂ CO ₃	5	86.4	94.2	7.8	7.8 = 9.03 86.4	5.8	83
11	30	25	7.2	Wet	CO ₂	1.00				3 min. in H ₂ CO ₃	3	86.4	93.6	7.2	7.2 = 8.34 86.4	6.4	68

Comparison of the results of this test with those of test 2, Table 1, in which pretreatment for 3 min. in carbonic acid took place and carbon dioxide was used for aeration instead of air, shows that they are of the same order of magnitude. Whether slight amounts of carbon dioxide in the air and water employed in the experiment had the conditioning effect on the magnesite as suggested can only be decided by further tests in which carbon dioxide is rigidly excluded. This may form the subject of a future investigation.

Test 7.—In test 4 (Table 1), as the addition of 0.85 lb. equivalent per ton sodium silicate to the cell feed improved the magnesium carbonate concentration, but 3.00 lb. equivalent per ton of sodium silicate under like conditions made no concentrate at all, it appears that small quantities of sodium silicate will improve the operation. Therefore in test 7 a sample was treated as in test 6, with one difference in procedure; that is, the addition of 0.5 lb. equivalent per ton of sodium silicate. The temperature was 23° C. The feed contained 84.25 per cent of total carbonates and the concentrate 92.0 per cent, an increase in total carbonates of magnesium and calcium of 8.75 per cent. The silica and hydrous silicates in the feed were 15.75 per cent and in the concentrate 8.0 per cent. Here, both the total carbonate recovery and the silicate reduction were not so pronounced as without the sodium silicate addition, although in smaller quantity than before.

Tests 8, 9, 10 and 11 were merely a continuation of the previous tests with other slight variations in conditions, in order to note their effects on the concentration. These are shown in Table 1.

SUMMARY OF RESULTS

1. It has been shown that magnesite can be successfully floated and its magnesium carbonate content increased in the concentrates.
2. Wet-grinding the samples of magnesite was the best method of conditioning the surface of particles, as a preliminary treatment for flotation.
3. Dry-ground samples may be conditioned with carbonic acid, and concentrating carried out by the use of carbon dioxide instead of air.
4. Oleic acid proved a successful collector during flotation, and helped in reducing the calcium carbonate, silica and hydrous silicate content of magnesite ore samples of the type used in the tests described.

Froth Flotation of Southern Barite Ores

By R. G. O'MEARA,* MEMBER, AND G. D. COE,† ASSOCIATE MEMBER A.I.M.E.

(New York Meeting, February, 1936)

PRIOR to the World War most of the barite used in the United States for manufacturing lithopone and barium chemicals was imported. Germany, by virtue of an abundance of high-grade ore and low labor costs, was in a position to dominate the world market. With the cessation of imports and with the war demand for barium chemicals, the barium chemical industry and the mining of barite became well established in the United States. This industry has continued to expand because of the increasing use of lithopone. In 1914 the sales of domestic crude barite was 52,919 tons and in 1934 it was 209,850 tons.

Barite deposits of economic importance occur in at least 20 states, but the principal producers in 1934 were Missouri, Georgia, Tennessee, South Carolina, Virginia, California and Arizona. The main mining centers are located near the crude barite markets of the three principal marketing districts—the Pacific Coast, the Midwestern area, and the Eastern area. The Pacific Coast is supplied entirely by California, Nevada, and Arizona. The Midwestern area uses most of the Missouri crude and a small amount from the southern states. The eastern market is the chief competitive area. Practically all of the imported crude is marketed there; also the bulk of the southern crude and ground barite. In addition, this area absorbs a small part of the Missouri crude and 80 per cent of the ground barite.

Besides the advantage of low delivered costs and purity of product, the imported barite has commanded a price differential because it is soft rather than hard, like many of the domestic ores. The inherent physical properties of the southern ores remain, but the purity can be controlled to a great extent by beneficiation. Simple washing or gravity concentration has sufficed in the past, but in many cases a more efficient process now becomes imperative to remove impurities not amenable to these methods. Certain deposits now being opened are of lower grade and require more precise milling conditions. In some ores, the kind of impurities has changed as the mining advanced.

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In 1934 the Southern Experiment Station of the U. S. Bureau of Mines in cooperation with the University of Alabama began an investigation of the beneficiation of southern barite ores. The investigation included gravity concentration and flotation. This report embodies the results of the preliminary flotation tests.

Five types of ores, which will be referred to as A, B, C, D and E, representative of southern barite deposits requiring beneficiation, have been examined. All the properties from which the samples were taken are either idle or handicapped in their operations by the lack of satisfactory concentration methods. The principal impurities are iron oxide, quartz, calcite, fluorspar, and metallic sulfides. Ore A is mainly barite and iron oxide; ore B, barite, iron oxide and fluorite; ore C, barite and quartz; ore D, barite, calcite and fluorite; and ore E, barite, fluorite and galena. Ore E contains only 32.45 per cent BaSO_4 , 5.3 per cent PbS and the balance is mainly fluorite.

Flotation tests have been made on these ores. Ore E is of such low grade that it was impossible to produce salable barite concentrates, therefore the results are not included. The prior selective flotation of the galena, however, offered no problem other than an additional middling loss of barite. Tests also have been made on a final concentrate from a mill treating ore of type A. After washing, screening, jigging and tabling, the concentrates from this mill still are too high in iron for certain uses.

The samples were ground in a porcelain mill with flint pebbles. Minus 65-mesh was fine enough for liberation of impurities and for flotation. On the high-iron mill product from ore A, a minus 35-mesh grind yielded a high-grade concentrate but the recovery was low, probably because of the inefficiency of the flotation cell on such coarse material. The samples were floated in distilled water in a 250-gram home-made machine of the Minerals Separation type. The rougher concentrates were cleaned in the same cell.

Two reagent charges were used; one for ores A and C (not containing fluorite) and the other for ores B and D (containing fluorite), as follows:

	POUNDS REAGENT PER TON FEED FOR ORES A AND C FOR ORES B AND D	
Sodium silicate.....	1.0	1.0
Sodium oleate.....	0.3	
Oleic acid.....	0.5	
Emulsol X ₁		0.5

The reagent charges have been interchanged, but the Emulsol charge appears less satisfactory for ores A and C and the oleic acid charge is wholly unsatisfactory for ores B and D.

With either set of reagents the ore was first conditioned about $\frac{1}{2}$ min. with sodium silicate. The other reagents were then added to the cell, conditioned a short period and the rougher concentrate was floated. One

TABLE 1.—*Flotation Results*

Product	Weight Per Cent	Assay, Per Cent		Distribution of	
		BaSO ₄	Fe ₂ O ₃	BaSO ₄	Fe ₂ O ₃
ORE A—ORE HIGH IN IRON. MINUS 65-MESH GRIND					
Cleaner concentrate.....	77.2	98.1	0.60	88.9	4.3
Middlings 1 and 2 combined.....	9.0	77.5	13.92	8.1	11.7
Rougher tailing.....	13.8	18.7	65.10	3.0	84.0
Feed (composite).....	100.0	85.3	10.70	100.0	100.0

Two cleaners with no reagent were used.

ORE A—MILL CONCENTRATE. MINUS 65-MESH GRIND					
Cleaner concentrate.....	87.8	98.8	0.07	93.9	3.5
Middling 2.....	4.4	94.7	1.16	4.5	2.9
Cleaner conc. and middling 2 combined.....	92.2	98.6	0.12	98.4	6.4
Middling 1.....	1.4	54.4	12.21	0.8	9.8
Rougher tailing.....	6.4	11.5	22.82	0.8	83.8
Feed (composite).....	100.0	92.4	1.74	100.0	100.0

Two cleaners with no reagents were used.

	Weight Per Cent	Assay, Per Cent			Distribution of		
		BaSO ₄	Fe ₂ O ₃	CaF ₂	BaSO ₄	Fe ₂ O ₃	CaF ₂
ORE B—ORE CONTAINING IRON OXIDE AND FLUORITE. MINUS 65-MESH GRIND							
Cleaner concentrate.....	68.9	96.9	0.43	0.13	76.8	5.9	1.6
Middling 2.....	4.0	95.0	1.73	0.62	4.4	1.4	0.4
Middling 1.....	10.2	87.8	4.48	4.11	10.3	9.1	7.5
Rougher tailing.....	16.9	43.9	24.85	30.12	8.5	83.6	90.5
Feed (composite).....	100.0	86.9	5.02	5.60	100.0	100.0	100.0

Two cleaners with no reagents were used.

	Weight Per Cent	Assay, Per Cent		Distribution of	
		BaSO ₄	SiO ₂	BaSO ₄	SiO ₂
ORE C—ORE HIGH IN SILICA. MINUS 65-MESH GRIND					
Cleaner concentrate.....	75.0	97.51	0.36	99.1	1.3
Middling.....	3.0	11.49	74.49	0.5	10.3
Rougher tailing.....	22.0	1.47	87.17	0.4	88.4
Feed (composite).....	100.0	73.80	21.67	100.0	100.0

One cleaner with 2.0 lb. sodium silicate per ton of original feed was used.

	Weight Per Cent	Assay, Per Cent			Distribution of		
		BaSO ₄	CaCO ₃	CaF ₂	BaSO ₄	CaCO ₃	CaF ₂
ORE D—ORE CONTAINING CALCITE AND FLUORITE. MINUS 65-MESH GRIND							
Cleaner concentrate.....	69.2	94.3	3.4	0.80	80.0	26.7	7.3
Middling.....	8.1	76.2	12.4	2.07	7.6	11.4	2.2
Rougher tailing.....	22.7	44.7	24.0	30.45	12.4	61.9	90.5
Feed (composite).....	100.0	81.6	8.8	7.63	100.0	100.0	100.0

One cleaner with 0.2 lb. sodium silicate per ton of original feed was used.

or two cleaners were required, in some cases with 0.2 to 2.0 lb. of sodium silicate per ton of original feed and in others, with no additional silicate. The flotation results are shown in Table 1.

The flotation of the high-iron ore (A) and the high-silica ore (C) is easily controlled and permits considerable leeway in manipulation. The ores containing fluorite are somewhat more sensitive. On these ores concentrates have been recovered that assayed higher in BaSO_4 and lower in CaCO_3 , and it is not improbable that the results can be improved. Various tests have been made to recover the fluorite after the barite, and to float the fluorspar first and then the barite. These results have not been superior to those of other investigators and are not included.

The results indicate that the southern barite ores of the type tested are amenable to flotation. The reduction of the laboratory tests to practice seems entirely possible in view of the successful work on other nonmetallics.

DISCUSSION

(W. H. Coghill presiding)

W. H. COGHILL,* Rolla, Mo.—The southeast Missouri deposits are entirely residual, extending over several square miles. Were the deposits where you found this material residual?

B. W. GANDRUD,† Tuscaloosa, Ala.—Most of them are, but the deposits in South Carolina, for instance, are vein deposits entirely, and in North Carolina there are vein deposits. In Tennessee and Alabama the deposits are residual, and I understand they are also in Georgia, although I have not visited the deposits in Georgia.

W. L. REMICK,‡ Hazleton, Pa.—Is there any material fine enough in the Georgia deposits to accept flotation?

B. W. GANDRUD.—I am not familiar with the Georgia deposits. It has been our impression that there is very little difficulty with the Georgia barites, and we have not collected samples from there, thinking that there was no trouble in producing a satisfactory commercial barite from those deposits.

R. W. SMITH,§ Atlanta, Ga.—The Georgia deposits are residual, and barite is found as nodules in residual clay. At one place some veins of barite show in the bedrock, which gives a clue to the origin of that residual barite, but the commercial deposits are residual and the operators have very little trouble.

P. M. TYLER,|| Washington, D. C.—Has much thought been given to the possibility of utilizing flotation as a means of bleaching barite? Can you improve the material by flotation?

B. W. GANDRUD.—The products that we have gotten have all been off-color barite; that is, concentrates. I suppose there would be some improvement if the iron oxide

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minerals were eliminated, but the problem of bleaching barites will have to be solved some other way than by flotation. We cannot hope to get a pure white product by flotation.

P. M. TYLER.—Am I right in thinking that barite is rather difficult to bleach?

B. W. GANDRUD.—We have not done any work on the bleaching of barites.

C. E. LOCKE,* Cambridge, Mass.—Have you undertaken to deal with the slime separately from the sand?

B. W. GANDRUD.—No, we did not separate the sand from the slime or the pulp after it had been ground to such a fineness, and the slimes did not seem to offer any difficulty.

J. R. CUDWORTH,† University, Ala. (written discussion).—There has been considerable interest in the Southern States in recent years in the possibilities of rendering some of the southern barite ores not now minable attractive to the consumer. In some places little or no effort has been made to beneficiate the low-grade deposits, while in others no changes have been made in the flowsheets to take advantage of improvements in ore-dressing practice.

In ores B and D in these tests, both of which contain fluorite, the recovery and grade was reasonably high for soap flotation. It is possible that these results could be improved by a longer period of conditioning. Certain other tests on similar ores have indicated that the period of conditioning is very important. The recovery and grade of ores A and C, in which the impurities are iron and silica, respectively, leave little to be desired. These results are excellent for soap flotation.

One of the outstanding points of the work is the successful use of one of the newer flotation reagents, Emulsol X₁ for the separation of barite from fluorite with lime or ron present.

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Government Prospecting for Phosphate in Florida

By P. V. ROUNDY* AND G. R. MANSFIELD,† MEMBER A.I.M.E.

(New York Meeting, February, 1937)

PUBLIC lands in Florida were first withdrawn from entry by President Taft on July 2, 1910, as a conservation measure because of their possible phosphate content. The reserve thus established was subsequently modified by further withdrawals and by restorations, so that on Jan. 1, 1934, it contained 66,796 acres, of which only a small part had been definitely classified. The reserved land was in scattered parcels, ranging from less than 40 acres to 320 or more, distributed in many counties. For administrative purposes and for final disposal of any of the numerous parcels, mineral classification of the land is legally necessary. In 1934 and 1935 the United States Geological Survey, under two successive grants from the Public Works Administration, prospected some 83 tracts of these lands in Polk County, comprising in all about 3300 acres, and six tracts comprising 240 acres in Citrus and Marion Counties. The tracts in Polk County were all in the so-called land-pebble field and the others in the hard-rock field. The Survey field party was under the direction of P. V. Roundy, from whose unpublished report the material in this paper is largely taken. The foreman of the prospecting crew was J. H. Wingate of Zephyrhills, Fla. The individual tracts were identified through the cooperation of the General Land Office, by transitman Hugh B. Crawford. This prospecting work fitted well the classification adopted for Public Works projects in general; it could not be done on any considerable scale under the Survey's usual appropriations; it gave employment to unemployed labor; and it may be expected ultimately to repay the Government in royalties more than the cost of the exploration.

COMMERCIAL PROSPECTING METHODS

Commercial methods of prospecting for phosphate in Florida have been briefly described by Mansfield.¹ In the hard-rock field, where the floor of the phosphate deposit is extremely irregular, it is customary to drill as

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¹ G. R. Mansfield: Phosphate Rock in 1923. U. S. Geol. Survey, Min. Res. of the United States for 1923 (1924) pt. II, 254-256.

many as 17 holes to the acre, or about 676 holes to a 40-acre tract, using generally a 50-ft. spacing. In the land-pebble field the deposit, though showing considerable variation in thickness and quality, is much more regular in its general mode of occurrence, therefore a wider spacing and smaller number of holes are usually satisfactory. For a 40-acre tract 16 holes ordinarily are sufficient. The equipment used differs in the two fields. In the hard-rock field, where large boulders of hard phosphate are encountered, a special type of drill is employed and usually a light, portable engine for power, though some drilling is done by hand. The overburden is first penetrated by a special tool or "cup," which "cups" a hole to the top of the phosphate deposit. In the land-pebble field a modified type of post-hole auger, described more fully later on, is employed. The hole is cased and all operations are by hand. Though the cost of prospecting hard-rock land on this basis may be as much as \$300 per acre, the cost of pebble prospecting may not exceed \$8 or \$10 per acre.

PURPOSE OF GOVERNMENT PROSPECTING

The purpose of Government prospecting is to obtain a basis for determining whether the Government should continue to hold the mineral rights in a given tract of withdrawn land or may properly restore said land for disposal under the nonmineral land laws. When the Government retains phosphate rights, individuals or corporations may lease the land or deposits involved and may produce phosphate therefrom on a royalty basis. Inasmuch as the primary concern of the Government is the mineral or nonmineral character of the land, its prospecting need not be so extensive or thorough as that of an operator that contemplates actual mining of the deposits. One to three or four holes may suffice to show whether justification exists for holding a 40-acre tract in a mineral reserve. Moreover, the Government is not necessarily concerned with the immediate exploitation of a deposit. It must consider future national needs and the probability that eventually deposits may be considered workable that are now thought too deep or of too low a grade.

CHARACTER OF TERRAIN

The surface features of the hard-rock and land-pebble fields are not greatly different. In both fields the surface of the country is generally fairly flat with interspersed lakes, swamps and wooded areas, though the hard-rock field is the more uneven and irregular. In the hard-rock field, however, the phosphate matrix is a residual deposit lying upon a limestone surface made highly irregular by the solution to which is also due the concentration and accumulation of the deposit. The overburden is sandy, clayey, and peaty material ranging in thickness from less than one foot to 50 ft. or more. The land-pebble deposits are parts of a fairly

evenly bedded formation that lies on a relatively smooth limestone floor beneath overburden of more uniform thickness. Prospecting work in either field, therefore, may fall in open ground, in forests, thick swamps, or even in ponds or lakes.

THE PEBBLE FIELD

Geology

The geology of the pebble field is summarized in Table 1, which shows the relation of the materials handled by the prospector to the recognized geological features of the region. This table is Roundy's interpretation of the materials passed through by the auger in the course of prospecting. Earlier and fuller accounts of the phosphate deposits and their geology are given by Matson,² Sellards,³ and Cooke and Mossom.⁴ Brief descriptions of the formations listed are given in ascending order.

Hard Bedrock.—The hard limestone beneath the phosphate deposits was seldom penetrated more than an inch or two in the drilling in Polk County, as all the pebble phosphate of workable quantity known in that region occurs above this rock. In some wells the hard bedrock reached may have been part of the Tampa limestone, which, though not exposed in Polk County, is said probably to underlie all upper Miocene or younger rocks in the county. It is thought, however, that in most of the wells the limestone called hard bedrock was some bed in the Hawthorn formation. Erosion and solution of the Hawthorn formation was probably uneven. It is therefore possible that certain wells in the same or adjacent 40-acre tracts may have reached different beds of this formation, which in the well logs were listed as "Top of hard bedrock."

Soft Bedrock.—The material classed in drilling as soft bedrock lies just above the hard bedrock. It usually contains a very small amount of phosphate and always a considerable amount of lime carbonate, as a powder, in clay, or as a soft plastic marl. It varies from very soft clay, easily penetrated by the auger, to tough harder material, possibly with thin limestone beds, which requires the use of a hard-rock drill. It may be sandy but does not contain distinct beds of pure sand. In some wells the soft bedrock may be unaltered soft beds of the Hawthorn formation; in some wells it may be composed of beds originally limy and hard but softened by subsequent leaching; in still other wells the soft bedrock may comprise both types of material. These beds appear not to have been otherwise disturbed or reworked since their deposition.

² G. C. Matson: The Phosphate Deposits of Florida. U. S. Geol. Survey *Bull.* 604 (1915).

³ E. H. Sellards: The Pebble Phosphates of Florida. Florida State Geol. Survey, 7th Ann. Rept. (1915) 25-116.

⁴ C. W. Cooke and S. Mossom: Geology of Florida. Florida State Geol. Survey, 20th Ann. Rept. (1929) 29-228. See also other State reports.

TABLE 1.—*Geologic Formations in the Land-pebble Field*

Prospector's Divisions		Lithologic Character	Formation	Series	System
Overburden (3½ to 98 ft.)		(a) Soil, sand, silt, clay, muck and peat		Recent	Quaternary
		(b) Sand and hardpan (a + b = 0 to 27 ft.)	This may be Recent or Pleistocene		
		Sand, sandy clay and clay (0 to 67 ft.)	Citronelle formation (?)	?	
		Black to dark greenish gray clay with very fine mica flakes (0 to 19 ft.)	Probably marginal phase of Bone Valley gravel		
Phosphate deposits		Phosphate pebbles in sand, sandy clay or clay matrix with some barren beds (0 to 24½ ft.)	Bone Valley gravel	Pliocene	
		Bed clay			
Barren beds		Sand, sandy clay, or clay. Locally a few minute, scattered phosphate pebbles. Locally lies on hard bedrock where soft bedrock is not recognized (0 to 19½ ft.)	May contain in lower part the equivalent of the soft bedrock		Tertiary
Soft bedrock		Soft clay has very strong lime reaction. Occasionally some sand, but not as distinct sand beds; usually shows slight amount of phosphate with chemical test (0 to 11 ft.)	Probably leached or partly leached undisturbed Hawthorn formation	Lower Miocene	
Hard bedrock		Hard bedded limestone, white to gray	Hawthorn formation		
			Tampa limestone		

Barren Beds.—In many places beds of clay, sandy clay and sand intervene between the present beds with phosphate pebbles and the soft or hard bedrock. These barren beds are discarded by the prospector. They may represent the lower part of the phosphate beds, the bed clay, or soft bedrock not otherwise recognized as such. The barren beds here and there contain considerable powdered phosphate, and in one well the top $1\frac{1}{2}$ ft. of those beds contained 24.8 per cent of tricalcium phosphate. In this well the upper part of the barren beds therefore belongs to the lower part of the phosphate deposit, probably the bed clay; the bottom part is evidently soft bedrock and the middle part may go either with the beds above or those below.

Phosphate Deposits.—The phosphate deposits are divided by the prospectors into two parts, the "phosphate beds" and the "bed clay." The bed clay, or lower part of the phosphate deposits, consists mainly of clay, sandy clay, or clayey sand, with or without abundant phosphate pebbles, and usually yields a carbonate reaction with acid. The contained pebbles tend to peel clean when the clay, as taken from the well, is broken in the hand. This tendency is so characteristic that it is used by the drillers as a means of recognizing the bed clay. In the main phosphate beds, the matrix sticks more noticeably to the pebbles. The character and number of pebbles in the bed clay vary as they do in the beds above, but in the bed clay the pebbles are larger and more abundant near the base, thus suggesting a basal deposit just above an unconformity. The bed clay may be absent or vary in thickness from a few inches to 5 ft. or more. In some wells it may have been present but not recognized. Usually the phosphate pebbles in this clay are darker in color than those in the main phosphate beds. The phosphate-pebble content of the bed clay varies from practically nothing to 20 per cent or more. In a few wells the bed clay was richer in pebbles than the beds above. The phosphate beds contain the richest and best grade of phosphate pebble. In the wells drilled during this investigation the phosphate beds ranged in thickness from nothing to 24 ft. 2 in. In this richest well, 4 ft. 4 in. of bed clay also contained phosphate pebble, giving a total for the well of $28\frac{1}{2}$ ft. of phosphate deposits. Some clayey beds contain few pebbles and some pebbly beds little clay. In addition, the deposits contain beds of sand and sandy clay. The contact with the overburden is ordinarily gradational as pebbles in the deposit become fewer. In some wells, however, the boundary between the two is more distinct and thus becomes both economic and geologic. The phosphate beds and bed clay are parts of the Bone Valley gravel of Pliocene age.

Overburden.—In the wells drilled during this investigation the depth of overburden ranged from $3\frac{1}{2}$ to 98 ft. The chief components are sand, sandy clay, clay, hardpan, silt, muck soil or sandy soil, and peat. Sand, sandy clay and clay comprise the greatest part of the overburden.

The sand is mostly quartz grains of different sizes but rather uniformly sized in a given bed. The beds vary in color and compactness. Beds of quicksand occur in different parts of the overburden from the bottom to near the top. Some sand beds tightly packed were difficult to drill but when loosened proved not to be cemented. Some thin beds were cemented but seldom caused trouble for the auger. Mixtures of sand and clay in which clay amounted to 10 per cent or somewhat more were called sandy clay by the drillers, but the change from sandy clay to clay, as the terms were used by the drillers, seemed to depend more upon the size of the sand grains than upon the actual percentage of clay. Most of the clay recorded as such in the overburden probably contained 10 per cent or more of sand. In some 20 wells drilled east of a fairly well defined line, the bottom of the overburden consisted of 2 to 19 ft. of dark, finely micaceous, tough clay that contained a considerable amount of carbonaceous matter but little or no quartz sand.

Hardpan was struck in about one-third of the wells. This material is sand cemented by iron and ranges in color from dark brown to nearly black. Some beds only an inch or less in thickness were called hardpan crusts by the drillers. The hardpan beds themselves averaged 2 ft. 1 in. in thickness and the maximum was 4 ft. Only two wells had more than one bed of hardpan, and these had two beds each. The depth of the hardpan beneath the surface ranged from 3 to 27 ft. Peat was found in only one well, where it was 15 ft. thick and very dark brown, almost black. Muck or silt was noted in about one-fifth of the wells but only at or near the top. Its greatest recorded thickness was $10\frac{1}{2}$ ft.

The age of the overburden is believed to range from Recent to upper Pliocene and the upper sand deposits beneath the top soil or muck appear to be material reworked from underlying deposits. The rather uniform relations of the beds of hardpan to probable ground-water levels, and their general absence beneath swamps or lakes, suggest that the cementation of these beds may have occurred in Quaternary time. The sands and sandy clays are thought to represent the Citronelle formation, though they differ in some respects from that formation as exposed at its type locality, and the dark micaceous clay is considered part of the Bone Valley gravel.

Location of Wells in Pebble Field

Selection of tracts for prospecting was made on the basis of records on file in the Geological Survey (Fig. 1). In locating the tracts, Hugh B. Crawford, of the General Land Office, kindly placed markers at corners or on the boundaries of the tracts and furnished information regarding means of access. The sites for drilling were located by the Geological Survey party by pacing, measuring with tape, or by intersection with alidade and plane table, using the Land Office markers as reference points.

Methods of Drilling, Pebble Field

Auger.—In drilling in Polk County a modified form of post-hole auger, about 4 in. in diameter and 1 ft. long, standard for the field, was used (Figs. 2 and 3). The top of the auger was threaded to receive the end of a standard $\frac{3}{4}$ -in. black iron pipe, which in pieces of convenient length (5 or 8 ft.) formed the stem or auger pipe, and a short crosspiece was screwed on the top length for a handle (Figs. 3 and 7). The auger was filled by turning the handle and was pulled out of the hole for cleaning after each filling. For beds too hard to be engaged by the auger, a drilling bit, chisel or curved chisel type, was screwed on to the pipe instead of the auger and worked up and down to break the hard material into pieces of suitable size to be received by the auger or sand pump.



FIG. 2.—COLLECTING PHOSPHATE SAMPLE FROM AUGER.

Casing.—The casing consisted of extra heavy " $4\frac{1}{2}$ -in." steel pipe, (actually 5 in. in outside diameter, $4\frac{1}{4}$ in. in inside diameter) weighing about 17 lb. per foot. The upper end was threaded on the inside and lower end on the outside, so that two pieces screwed together would form smooth joints both outside and inside. The threads were cut square, about five to the inch. The bottom, or lower section, called the "gun," was threaded only at the upper end. The lower end was notched with large sawlike teeth, the ends of which were bent slightly outward, so that this section of material forced into the casing as it was lowered would be almost exactly 5 in. in diameter. A casing clamp made of 4 by 4 or 4 by 5-in. timber 5 or 6 ft. long was fastened to the top of the casing, as shown in Fig. 4.

tended to the top of the casing, as shown in Fig. 4.

Sand Pump.—The sand pump, a common driller's tool, consisted of a steel pipe, $3\frac{1}{4}$ in. in outer diameter, 42 in. long with a flapper valve at the bottom and a fixed bail at the top, to which was attached a long rope (Fig. 5).

Drilling Operations.—At the desired site a vertical auger hole was bored to a depth of about 6 ft. (or less if the hole tended to cave) and the gun inserted (Figs. 3 and 6). Men pushing back and forth at opposite ends of

the casing clamp twisted the casing in the ground, causing it to sink of its own weight (Fig. 4). When in clay or other material that did not cave, the auger was kept even with or ahead of the casing, but in material that caved the casing was kept ahead of the auger. In quicksand or soft material that could not be brought up with the auger, the sand pump was used. Repeated raising and dropping of the pump a few inches filled it with slush and it was emptied from time to time. By "working the casing" as the pump was used, the casing could be kept well ahead of the pump (Fig. 8). As an aid in steadying the auger stem a "lean pole"—a slender pole 25 to 35 ft. long, forked at the top—was used. It was



FIG. 3.

FIG. 3.—STARTING A WELL WITH THE AUGER.



FIG. 4.

FIG. 4.—THE CASING CLAMP SINKS THE CASING AND SERVES AS A PLATFORM FOR THE DRILLERS

inserted obliquely about 4 ft. in the ground at one side of the well, so that its fork was directly above the well (Fig. 7). The auger stem up to lengths of 35 ft. slid and rested in this fork as it was being pulled and the auger cleaned. When the hole was deeper than 35 ft. the auger stem was "cut" or unscrewed into sections about 25 ft. long and each section was allowed to stand in the fork as more auger stem was pulled. In some places a tripod was used instead of the lean pole. It consisted of three long slender cypress poles fastened together at the top with a heavy bolt, which also passed through the ends of a U-shaped iron loop (Fig. 8). A convenient tree could be used in some places instead of the pole or tripod.

In drilling, three men usually worked on a well until a depth of about 50 or 60 ft. was reached, then a fourth man was added. Occasionally five men were needed when drilling was unusually difficult. A record of the beds passed through was made by examining the contents of the auger

and the depths were obtained by measuring the auger stem and subtracting the length above ground before pulling the auger from the well. When phosphate was found the casing was sunk slightly below the bottom of the drilled hole and the sand pump was used to clean out the loose overburden and the water mixed with it. The phosphate was then brought up with the auger and placed in bags (Fig. 2). The phosphate that was mixed in the water by the auger was recovered with the sand pump and the mixture was poured into bags, which were placed in holes dug in the ground. This allowed the excess water to drain out, leaving most of the phosphate



FIG. 5.—SAND PUMP READY FOR USE.

behind in the bag. When in phosphate the casing was kept ahead of the auger, so as to obtain a sample originally 5 in. in diameter. Otherwise the sample would contain an unduly large proportion of the sandy and soft parts of the beds.

The deepest hole drilled by these methods was 109½ ft. deep. So far as known to the writers and the prospector this was the deepest well ever drilled in the pebble phosphate field using 5-in. casing and man power alone. The casing in this well weighed about 1800 lb.

When a well was completed the casing was pulled by looping a log chain around it and over the end of a long log lever, as shown in Fig. 9. Two or more men bore down on the end of the lever while other men loosened the casing in the ground by alternately pushing and pulling on the ends of the casing clamps. The time taken to pull and unscrew the casing by this method depended

more on the nature of the overburden than upon the depth. When quicksand was found directly below a bed of hardpan, the pulling of the casing was a long, hard process. For a few wells, when this condition obtained, the operation took two or more days per well. In some wells the pulling of the casing took only an hour, or less. In No. 12, the deepest well drilled, the casing was pulled in 2 hr. and 40 min.

Wells were drilled under different types of surface conditions, such as open country, improved property, thick-grown swamps, flooded areas, and even in lakes (Fig. 10).

Sampling Pebble Phosphate

Treatment of Crude Phosphate.—Crude phosphate, or simply “crude,” consists of different kinds of pebbles, mainly phosphatic, mixed with sand and clay. For purposes of classification work it is considered as consisting of three parts—pebble, matrix and moisture. The sample collected in bags was allowed to stand until the excess water obtained by using the sand pump had drained off. The material was then weighed (gross crude weight) and the weight of the containing bags subtracted (crude weight).



FIG. 6.

FIG. 7.

FIG. 6.—INSERTING “GUN” IN WELL.

FIG. 7.—LEAN POLE USED FOR STEADYING AUGER PIPE.

The sample was washed in a box of which the bottom was made of sheet steel punched with slots $\frac{1}{2}$ in. long and $\frac{1}{32}$ in. wide. All the material retained by this screen was called pebble. The material passed through constituted the matrix, and the wash water was saved. The pebble was dried in a sheet-iron pan over a fire and weighed as “dry pebble.” Next the matrix was dried and the wash water evaporated in similar fashion and weighed as “dry matrix.”

Moisture.—The crude weight minus the combined weight of the dry pebble and dry matrix equals the weight of the contained moisture, composed of interstitial adhering water and water contained in the pebbles. The drying is not intended to calcine the material, only to remove the uncombined water, which, reduced to a percentage of crude weight, averaged about 33 per cent.



FIG. 8.—WORKING THE CASING DOWN, A TRIPOD BEING USED INSTEAD OF A LEAN POLE.



FIG. 9.—METHOD OF PULLING CASING.

Pebbles.—The pebbles have a minimum diameter of $\frac{1}{32}$ in. Usually they are less than $\frac{1}{2}$ in. but larger pebbles are found. Some too large to be recovered with the auger had first to be broken with the drill. Although most of the pebbles consist of phosphatic material, some contain no phosphate. Pebbles of flint, chert, limestone, quartz, silicified wood, calcareous fossils, and other substances are not uncommon. The phosphatic pebbles differ in composition, shape, size and color. Some apparently are rather pure, others contain iron, clayey or carbonaceous material,



FIG. 10.—DRILLING TO 103½ FEET BELOW WATER SURFACE IN AREA FLOODED BY DAMMING WITHLACOOCHIE RIVER (HARD-ROCK FIELD).

or other substances. There are many fish remains, especially sharks' teeth. As a rule, darker pebbles are more abundant in the lower part of a deposit but locally they are abundant in other parts as well.

Matrix.—The components of the matrix are similar to those of the pebbles but in different proportions, and in the form either of small grains or powdered material. Clay was usually present and many minute sharks' teeth were noted. Fine quartz sand predominated.

Treatment of Dry Pebble and Matrix.—The crude weight of the samples as obtained from a single well ranged from very little to as much as 542 lb. After washing and drying, large samples were reduced by repeated mixing and quartering to weights of 10 to 15 lb. each. Through the courtesy of the Southern Phosphate Corporation these samples were crushed and pulped at its laboratory near Bartow. Analyses to determine the $\text{Ca}_3\text{P}_2\text{O}_8$

(bone phosphate of lime) and the $(\text{AlFe})_2\text{O}_3$ content of each sample were made at the laboratories of the U. S. Geological Survey in Washington.

Estimating Tonnage

In previous phosphate prospecting in Florida it has been the practice to determine the tonnage of pebbles per acre and the percentages of tricalcium phosphate and of the oxides of iron and alumina in the pebbles. In view of the present use of flotation methods of phosphate recovery in Florida, it seemed desirable to include the matrix as well as the pebbles in sampling the deposits and estimating their tonnage. Beginning with a few wells, this practice was finally extended to all wells subsequently drilled. Two general methods of computing tonnage of pebbles per acre from prospecting data are used in Florida—the Pratt method, developed by Dr. J. H. Pratt, and the Meade method, devised by H. L. Meade. The Meade method is based on the weight of pebble obtained from a well, the core of which had a perfectly uniform diameter, and the relation of the area of this hole to an acre.

Pratt Method.—The Pratt method, which is the more widely used, is based on the thickness of the phosphate bed, its percentage of contained pebble and the specific gravity of the deposit. In practice, the weight of a cubic foot of crude is used instead of the specific gravity. For much of the Polk County deposits this weight is 125 lb. Thus:

$$\frac{\text{Sq. ft. per acre (43,560)} \times 125}{\text{long ton (2240 lb.)}} = 2430.8 \text{ or (omitting the 0.8) } =$$

$$2430 = \text{constant } 2430 \times \text{thickness (ft.)} \times \text{percentage of pebble} = \text{tons of pebble per acre.}$$

Since the weight of crude per cubic foot actually varies from place to place, correction must be applied in each case to the result obtained. This is easily done, for tables computed by Roundy show that a change of one pound in weight of a cubic foot of crude changes the total tonnage $\frac{1}{5}$ ton per foot of crude for 1 per cent of pebble.

For example, a 15-ft. bed of crude weighing 125 lb. per cu. ft. and containing 12 per cent pebble indicates $2430 \times 0.12 \times 15$, or 4374 tons of pebble per acre. However, if the crude weighed only 121 lb., 4 lb. less than standard, the correction is $(-4) \times \frac{1}{5} \times 15 \times 12$, or -144 tons. The amount of pebble would then be $4374 - 144$, or 4230 tons per acre. Ordinarily the bottom sample weighed slightly less than the top sample.

In general practice the thickness of the deposit is taken to the nearest half foot. In the present work it was recorded to the nearest inch and this figure was used. By the method used to estimate tonnage, a variation of 3 in. in thickness of a deposit gives a variation of 60 tons per acre for 10 per cent pebble content, or 243 tons per acre for 40 per cent pebble. The tonnage of matrix was computed separately for each well in the same manner as

the pebble content, and the sum of phosphate in pebble and in matrix was taken as the total tonnage of tricalcium phosphate per acre for a given tract.

Estimating tonnage per acre from a single well involves very large extrapolation from available data. Actual knowledge is confined to the area of the hole from which the samples were taken and the ratio of this area to an acre is about 1:300,000. However, a deposit in a given area is so generally uniform that in commercial practice 16 holes ordinarily suffice for 40 acres. As already explained, the Government's needs can be met by using fewer holes.

HARD-ROCK FIELD

Five wells were drilled in Marion County in the hard-rock field and six in Citrus County.

Geological Conditions.—The rocks of the areas prospected consist of Recent deposits of wind-blown sand, stream and swamp deposits, soil, etc., above the Pliocene Alachua formation. The Alachua formation consists of clays, sandy clays, and some sand beds, with phosphate in its lower part. The clays and sandy clays are of different colors with tints of red and yellow predominating. The colors are much brighter than those of the equivalent Bone Valley gravel in Polk County. The lower part of the Alachua formation contains phosphatic boulders of different sizes up to masses weighing several tons. The thickness and distribution of these deposits are very uneven and irregular.

The Alachua is regarded by Cooke and Mossom as a residual formation derived chiefly from the Hawthorn formation, which originally covered the hard-rock phosphate area. The phosphate granules in the Hawthorn were probably dissolved and redeposited as hard-rock phosphate in the lower part of the less soluble residual clays and sands now forming the Alachua formation and in the top of the underlying limestone.

The Tampa limestone, which underlies the Hawthorn formation in Polk County, and the Suwannee limestone, formerly included in the Tampa (basal part), were probably eroded from most of the area prospected. However, it is believed that the limestone found below the phosphate in one well in the south part of Citrus County was the Suwannee. With this exception the Alachua formation rests on the rough, irregular surface of the Ocala limestone.

Drilling Methods.—Drilling here requires a hard-rock type of drill and a method of driving the casing. The usual method is to use a heavy 2-in. casing and a fishtail type of drill attached to the end of a $\frac{3}{4}$ -in. pipe. The drill bit has a hole in each side of the blade and a check valve inside, so as to force the drilled rock into the drill pipe and retain it as a sample. A heavy block is used to drive the casing. With men trained in the Polk County work, this method at first proved slow and unsatisfactory.

Later it was modified by using the pebble-field methods until the top of the hard rock was reached. Then the 2-in. casing was inserted in the larger casing and drilling continued with the hard-rock drill. This method proved faster and the crews were able to reach greater depth.

Spacing of Holes.—In mining hard-rock deposits, three wells spread over a rich deposit in a 40-acre tract may fail to show even a trace of phosphate. In commercial practice, as stated, test wells are drilled systematically 50 or 100 ft. apart over the area and some additional holes midway between these. For the Government work such a spacing was impracticable, and it was necessary to rely on the findings obtained in three or four holes to a 40-acre tract. Because of the nature of such deposits, relatively greater significance from the Government's viewpoint would attach to the finding of even poor showings of phosphate than would be true in the pebble field. Thus the finding of phosphate in more than traces in any well would seem to justify retention of mineral rights by the Government, pending the issuance of prospecting permits and more intensive exploration by interested private agencies.

Sampling.—Here, as in the pebble field, an effort was made to save the entire sample. The material forced out of the drill stem was collected in a wash tub and the finer separated from the coarser particles by sifting. These fractions were weighed and analyzed separately but the results were combined in relative proportions in computing the total tonnage.

Tonnage Computation.—The method of computation used was essentially that described for the pebble field, except that the results were figured as tons of contained tricalcium phosphate per acre, rather than as tons of rock.

Mining Practice in the Florida Pebble Phosphate Field

BY CHESTER A. FULTON,* MEMBER A.I.M.E.

(New York Meeting, February, 1936)

IN Polk County, Florida, the mining of raw phosphates began some 50 years ago with dredging operations on the Peace River, and in other near-by places by removal of shallow overburden with negroes and wheelbarrows, or with mules and scrapers, and the excavation and screening of deposits found close to the surface. There were many individual operations of comparatively short life. Later steam shovels removed the overburden, and still later hydraulic methods were substituted for the steam shovel. Within the last 10 years large electrically operated draglines have almost entirely replaced hydraulic mining for overburden removal. The matrix is still mined by hydraulic giants and pumped to the washer or recovery plant.

The many small operations have long since disappeared and the most desirable phosphate lands have gradually been acquired through purchase or merger by the seven companies that are operating in the field today. These companies are producing and selling about 2,000,000 long tons of phosphate per annum, or about two-thirds of what the industry likes to think is the normal output, with 1929 and 1930 in mind. To produce these 2,000,000 long tons of raw phosphate, they mine and treat about 1,000,000 tons of tailings from older and less efficient operations, about 11,000,000 tons of matrix, and remove from above this matrix some 22,000,000 tons of overburden.

The mining practice that has made possible the excavation and handling of 34 million tons of material to win but 2 million tons of merchantable product, selling for probably an average of only \$4 per long ton when dried, should be of interest to the mining profession, though little has been written of it.

The Florida Pebble Field, as it has come to be known, comprises an area (Fig. 1) within a radius of about 18 to 20 miles centering around Mulberry in Polk County, a small town 30 miles east of Tampa, Florida's largest port on the Gulf coast.

The geology¹ of these phosphate pebble deposits may be described briefly. A fairly pure limestone was altered to a secondary apatite.

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¹ E. H. Sellards: Origin, Mining and Preparation of Phosphate Rock. *Trans. A.I.M.E.* (1915) 50, 907.

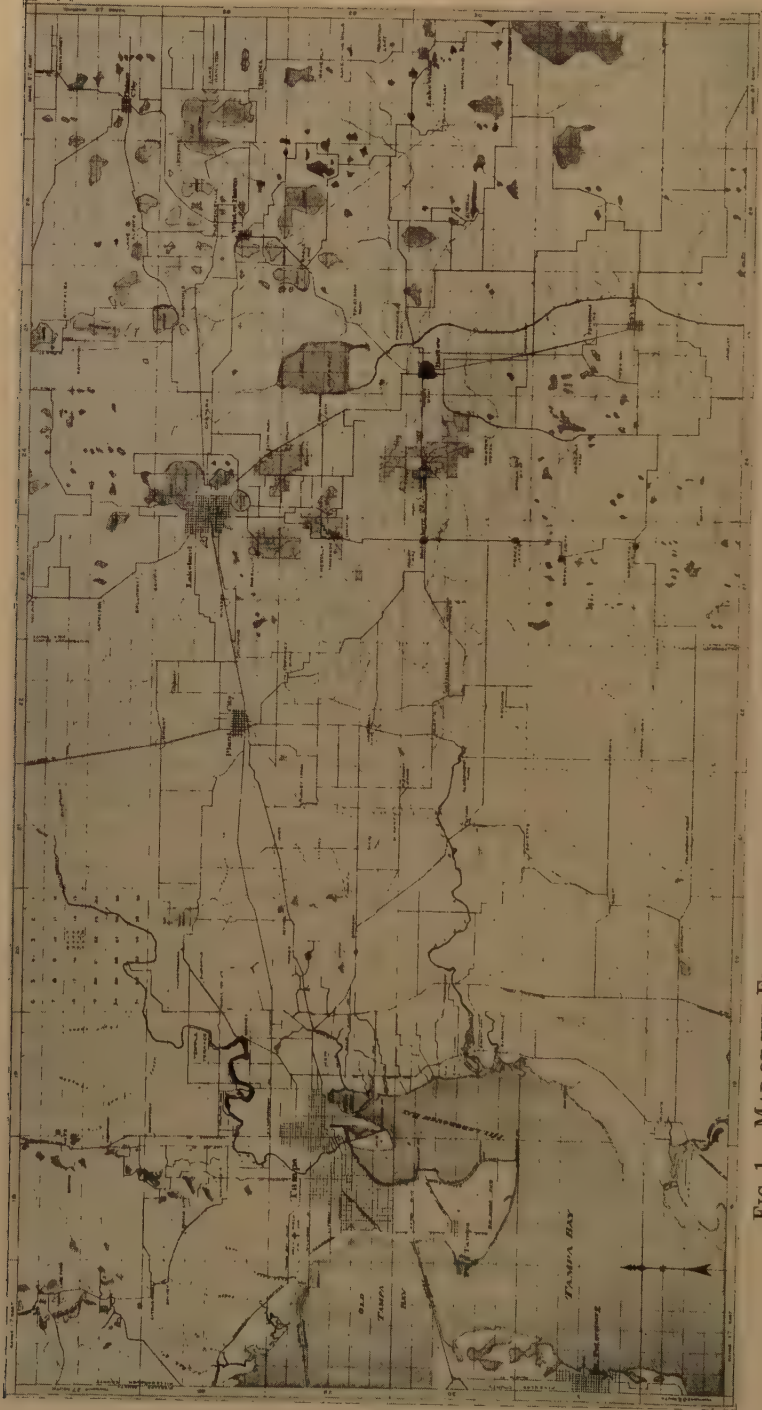


FIG. 1.—MAP OF THE FLORIDA PEBBLE PHOSPHATE DISTRICT, SHOWING ITS RELATION TO TAMPA BAY.

Subsequently this eroded, and together with other products of erosion such as sand and clay, accumulated in basins, resulting in the beds, which are today called the "matrix." Over this matrix has been more recently deposited an overburden of sand and clay, accompanied in some parts of the field by thin strata of sand rock. The matrix varies in thickness from nothing to 40 ft. Profitable mining usually requires that the matrix bed be at least 6 ft. thick, and generally it is not thicker than 25 ft. In exceptional instances, the overburden is only a few feet deep, usually 15 to 25, while 50 ft. is close to the limit for profitable operation.

While this paper describes the practice at a single mine with which the writer is familiar, this practice is typical of that prevailing in the field.

PROSPECTING

Phosphate lands are acquired only after they have been prospected by auger drilling. Until about 10 years ago most lands were bought on the prospecting of Dr. J. H. Pratt, a chemist of Bartow, Florida, whose work was excellent and whose integrity was beyond reproach. At that time the matrix treatment was a simple one that extracted only from 50 to 60 per cent of the phosphate by scrubbing and screening on about 14 mesh, coarser than which there was no sand in the matrix to dilute the phosphate, and the tailings, often containing more phosphate than that recovered, were sent to the "debris pile," the local term for a tailings dump. Today, with from 85 to 90 per cent of the phosphate recovered, Dr. Pratt's maps must be corrected by careful guessing based on partial re-prospecting.

A 4½-in. auger hole is driven by hand vertically through the overburden and matrix. Such holes are driven at a rate of about 5 ft. per hour, and cost about 30¢ per foot of hole. The entire core is recovered, weighed and sampled. Complete data are calculated from screen analyses, chemical assays and recovery tests. A complete prospecting is considered to require 16 holes on each 40-acre tract (Fig. 2). A prospect map shows the following data at each hole:

3051		2052
75.13	21	76.46
	●	
1.62	9	2.12
6.47		5.48

which means that at this hole there is 21 ft. of overburden and 9 ft. of matrix below the overburden; 3051 long tons (dry basis) of phosphate rock per acre coarser than 14 mesh, assaying 75.13 per cent bone phosphate of lime (tricalcium phosphate, $\text{Ca}_3(\text{PO}_4)_2$), 1.62 per cent I.&A. ($\text{Fe}_2\text{O}_3 + \text{Al}_2\text{O}_3$), 6.47 per cent insoluble (silica and insoluble silicates); and, in addition, finer than 14 mesh, 2052 long tons per acre, assaying 76.46 per cent B.P.L., 2.12 per cent I.&A., 5.48 per cent insoluble. From

these data are calculated the cubic yards of overburden and of matrix per ton of recoverable phosphate, as well as the total tonnages by grades of the phosphate in the property.

Thus the mining company before mining an area of phosphate land can plan in advance the entire operation, location of recovery plants, tailings disposal, drainage and water circulation, and the direction and location of the successive mining pits or "cuts."

PLANNING OPERATIONS

On a tract of land, say 2000 acres, containing some 6,000,000 tons of recoverable phosphate, with a production of 1500 tons per 24-hr. day, or a possible 450,000 tons per annum, mining operations will be carried on, theoretically, for a period of about 13 years. Practically it will take probably 25 years to mine out such an area because production is dependent on sales, and sales are dependent on the varying economic conditions of the farmer in the older agricultural countries of the world. Yearly production in a given area is also dependent on the grades of phosphate pebble that can be produced in that area, for all grades are in demand, while a tract of land is generally predominantly low, medium or high grade, though containing some tonnages of all grades. Hence most of the mining companies are compelled to have more than one mining operation and usually none is operated continuously for the entire week.

The phosphate matrix is mined with hydraulic giants or "guns," and the slurry is pumped to the "washer" or recovery plant. There is an economic limit to the distance the matrix can be pumped and the usual practice is to consider this limit reached when the operation requires two booster pumps in the line in addition to the pump in the pit. Roughly, this limits mining to within one mile of the washer. Beyond this limit it becomes less expensive to dismantle the washer and rebuild it in a new location. Generally speaking, in a high-grade deposit, one washer will have a life of 5 to 10 years and will produce between one and two million tons of phosphate. A 2000-acre tract, therefore, probably will be divided into three or four washer locations. In the planning of these locations there must be considered, and provided for, adequate room for tailings disposal, and gradual settling out of the clay slime from the tailings water, so that this water may be used again when finally it returns to the sump or pond at which the pumping station is located.

Tailings Disposal and Water Circulation

For the disposal of the tailings and the circulation of water a very elaborate and ever-changing system of dams and ditches is provided. At the start of mining operations on a new tract of land, tailings may flow by gravity in a wooden flume approximately 2 ft. 6 in. wide by 2 ft. high

on the sides, with a grade of about $\frac{1}{8}$ in. to the foot. This flume conducts the tailings from the washer to a broad, ever-widening dump located on unminable land and surrounded by an earth dam, for stability. The drainage of water from this dump, a flow of between 8,000 and 10,000 gal. per minute, is then conducted by dams and ditches in a circuitous route to return to the pond at the pumping station. Where barren areas do not exist near a new washer it becomes necessary to pump the tailings to the nearest unminable area, and this condition may have an important bearing on the location of the washer. But after mining operations have been under way for a year or more the ever-increasing mined-out area furnishes an excellent means for disposing of tailings and for circulating the water.

Normally this section of Florida has sufficient rainfall to offset the evaporation of the water supply, which has thus been spread out over considerable area, but there is a dry period during February, March and April when precipitation is at a minimum and the supply of water diminishes through evaporation, requiring the operation of deep-well pumps to replenish the loss.

Some 800 to 900 ft. below surface there is a bed of cavernous limestone (Ocala) which when tapped by a deep well will yield 5000 gal. or more per minute. Such a well costs from \$8,000 to \$15,000 fully equipped and is drilled in a manner similar to the drilling of an oil well. One typical well starts with a 26-in. casing and ends with a 16-in. It is equipped with a No. 20 three-stage Worthington deep-well Coniflo pump, capacity 4500 gal. per minute against a discharge head of 137 ft., directly connected to a General Electric slip-ring, variable-speed, vertical motor, 250 hp., 2300 volt. The pump is set 120 ft. below the surface, with 20 ft. of 14-in. suction pipe; the motor is set up on foundations at the mouth of the well. Such a pump is operated when necessary, on the average of about 300 hr. per annum.

During the periods of heavy rainfall it becomes necessary to slough off water through well constructed weirs into the streams forming the natural drainage of the country. As the circulating water contains very finely divided or colloidal matter, it is slightly "milky" in appearance and care must be exercised in its release, to avoid complaints and suits for polluting the waters traversing agricultural lands or even lands of little economic value but in the hands of mercenary owners.

As previously indicated, the return water in the circulation system finally reaches a sump or pond alongside of which is the pumping station, consisting of three pumps. The main pressure pump furnishing water to the guns in the pit is a two-stage Allis-Chalmers centrifugal directly connected to a 750-hp. synchronous motor, 1200 r.p.m., 2300 volts. This unit has a capacity of 5000 gal. per minute against a 476-ft. head. It is operated to supply about 4000 gal. per minute at a pressure of 200 lb.

per square inch at the nozzles of the guns. Usually this means about 220 lb. at the pump. Two other centrifugal pumps furnish water for ore-dressing purposes in the washer. One of these is a single-stage Allis-Chalmers pump directly connected to a 75-hp., 870 r.p.m., 2300-volt G.E. induction motor with a capacity of 5000 gal. per minute against a 50-ft. head. The other is also an Allis-Chalmers single-stage pump directly connected to a 150-hp., 1760 r.p.m., 2300-volt Allis-Chalmers induction motor and has a capacity of 2000 gal. per minute against a 180-ft. head.

The ditches and dams controlling the settling area and the circulation of the water are dug or built up by means of a $\frac{3}{4}$ -yd. Bucyrus Diesel dragline. The machine carries a 45-ft. boom and travels on caterpillars. In very soft or swampy land timber mats are laid down, 4 ft. wide by 16 ft. long. The machine places its own mats. It can excavate about 500 cu. yd. of earth in an 8-hr. shift; or, let us say, in that time it can dig a ditch 20 ft. wide at the top, 8 or 10 ft. wide at the bottom, 10 ft. deep and 90 ft. long. Dams usually are 12 to 15 ft. high and 25 to 30 ft. wide at the base, the size and height varying with the natural rise and fall of the terrain. Earth excavation with a Diesel dragline of this size can be done for an out-of-pocket cost of about $3\frac{1}{4}\epsilon$ per cubic yard.

REMOVAL OF OVERBURDEN

The site for the washer having been selected, the railroad company builds to it from the nearest line; the phosphate field being well serviced by main lines or branch lines of either the Atlantic Coast Line Railroad or the Seaboard Air Line Railway. Suitable sidings and tail tracks are built at the washer site, for there will be days when as many as 50 fully loaded "wet-rock" cars each holding 35 to 40 long tons (dry basis) of wet phosphate will move out to the drying plant. The washer is then erected. It is a sizable ore-dressing plant capable of treating about 9000 long tons of matrix per 24 hr., from which it will produce, say, 1500 to 2000 tons of raw phosphates.

Mining operations then begin on one side of a north-south or east-west line bisecting the washer, for Florida property lines run parallel to ranges and townships, typical of what once was the public domain (Fig. 2). The first operation is overburden removal, which is closely followed by hydraulic mining of the matrix.

The overburden is removed from the top of the matrix by large electric draglines; commencing, for example, $\frac{1}{2}$ mile or more to the west of the washer and progressing northward for $\frac{1}{2}$ mile with a cut about 210 ft. wide. The overburden is dumped by the dragline into the mined-out cut alongside. The overburden from a "pioneer" cut, which is the first cut on the edge of the property, must be piled temporarily on what will later be the second cut, and when that is mined it must be handled

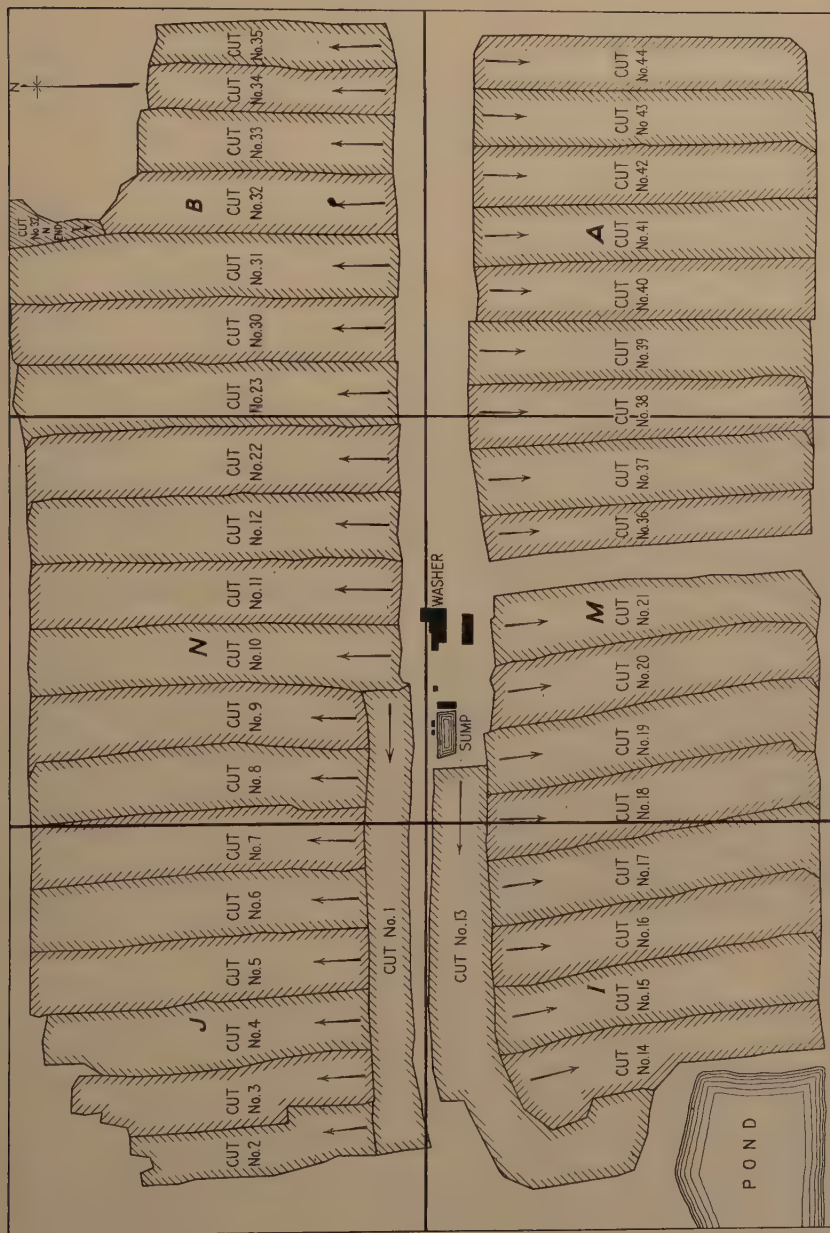


FIG. 2.—PROGRESS MAP SHOWING SUCCESSIVE MINING CUTS.

again, together with the overburden from the second cut, and dumped in the first cut. To start the second cut the dragline and all mining equipment must again be moved to the east-west line and mining proceeds northward again. In this way pipe lines are kept out of the mined-out cut where overburden will be dumped.

The overburden, usually a mixture of sand and clay with now and then thin layers of sand rock, is removed by a type 320-B electric dragline (Fig. 3) made by the Bucyrus-Erie Co. This huge machine is of all steel construction, about 47 ft. long, 40 ft. wide and 36 ft. high to the top of the



FIG. 3.—MINING PIT, SHOWING OVERBURDEN SPOIL BANK, 8-YARD DRAGLINE PIT CAR, WELL AND HYDRAULIC GUNS IN OPERATION.

operating cab. It is equipped with a cantilever boom hinged at its lower end to the revolving frame, about 15 ft. above ground. The boom is 138 ft. long, with its outer end suspended usually about 95 ft. in the air, and from it hangs an 8-yd. Bucyrus bucket. The weight of such a dragline is 1,100,000 lb. In 1928, the cost of such a machine erected and ready to operate was \$177,000.

The machine is supported by and travels on four sets of double caterpillar treads. Each tread is 17 ft. long by 3 ft. wide, making a combined supporting surface of 408 sq. ft. Each of the four sets of caterpillars can be operated on high or low speed individually, and in this way the machine is steered when traveling. Above the caterpillars is the bottom frame carrying a circular track on which the revolving frame rotates on a ring of rollers 26 ft. in diameter. The upper part of the machine can

rotate in either direction through one or more complete circles. The revolving frame supports all of the electrical and mechanical equipment necessary to operate and control this huge machine in its digging operations. This is all housed in a steel cab 20 ft. wide by 46 ft. long.

Space does not permit of a detailed description of this dragline's equipment. It consists of drag cable drum and hoist cable drum, a 105-hp. swing motor and two 150-hp. hoist motors, all direct current, 250 volts. There is a motor-generator set of 435 kva., 2200 volts, three-phase, 60-cycle synchronous motor direct connected to three generators; viz., a 75-kw., 250-volt d.c. generator to drive the swing motor, a 275-kw., 500-volt d.c. generator to drive the two hoist motors connected in series, and a 20 kw., 125-volt d.c. generator to furnish exciting current, control current and field current. There are also a 20-hp. air-compressor unit and a 2-hp. blower unit for cooling brake drums. All electric equipment is of General Electric manufacture. The type of control is known as the "Ward-Leonard" and gives a smooth and almost stepless speed range from standstill to full speed.

The machine requires from 40 to 65 sec. per bucket of overburden, with an average of 52 sec. per cycle, and takes out in each bucket an average of $8\frac{9}{10}$ cubic yards, equal to about 600 cu. yd. per hour.

About three 8-hr. shifts are required to strip one move, 60 ft., of overburden from the matrix; working on a cut 210 ft. wide in overburden of 20 to 25 ft. deep.

The dragline loses about 5 minutes of digging time on each move back of 60 ft., this occurring on an average of once each 24 operating hours; the machine will move back from the finish of one cut to the beginning of the next cut, a distance of $\frac{1}{2}$ mile in 5 hr., when over ordinary solid ground, and in addition will handle the moving of the pit car and mining equipment.

The crew on each 8-hr. shift consists of three men; an operator, an oiler and a ground man. The operator, being in charge of each shift, does all of the operating, superintends the repairs, and orders supplies and repair parts. The oiler greases the machine, helps with the repairs, and while the machine is moving is responsible for ground operations, such as guiding the machine, condition of ground to be moved over, and anything that cannot be seen by the operator from above. The ground man helps with the repairing, cleans the machine and helps with moving the machine.

The bucket in use is a Bucyrus-Erie, capacity 8 cu. yd., weighing about 17,300 lb.; it has been off the machine but once in about seven years, and then for major repairs. More than 13,000,000 cu. yd. of material has been moved by this bucket.

The cables in use on this machine are Leschen Hercules Red Strand improved plough steel Lang lay as follows:

One 1-in., 6 by 19, wire-rope center cable for raising and lowering boom, 1680 ft. long, with an average life of 3 to 5 years, depending on operating time of the dragline.

Two 2-in., 6 by 19, wire-rope center cables, for safety cables on boom, 125 ft. long, with an average life of 10 years.

One 2-in., 6 by 19, wire-rope center cable, 220 ft. long, used as a drag cable for pulling the bucket into the dirt. This cable has an average life for digging 326,000 cu. yd. of overburden and matrix.

One 1¼ in., 6 by 19, wire-rope center cable, 550 ft. long, used as a hoisting cable for raising and lowering the bucket, with an average life for digging 727,000 cu. yd. of overburden and matrix.

Ten 1½-in., 6 by 19, wire-rope center cables, 65 ft. long, used as sway cables along each side of the boom and having an average life of 10 years.

Three extra flexible, single conductor copper cables, 1000 ft. long, used as ground cables to conduct the 2300 volts for operation of the machine. These cables are constructed as follows: Over the No. 0 flexible copper center is ⅜ in. of high grade rubber insulation and over this is a hard loom braid for protection of the rubber insulation. They have an average life of 2½ years.

After stripping the overburden from the matrix, the dragline also digs 30 to 40 per cent of the matrix from the spoil bank side of the pit, dumping this on top of the undisturbed matrix bed. There are several advantages resulting from this additional work of the dragline: (1) The dragline can strip a cut 25 ft. wider with an increase of about 5 per cent in the yards moved per hour, because overburden can be disposed of in the space from which this matrix was removed with less movement of the dragline boom, requiring a much shorter digging cycle; (2) this uplifting and moving over of the matrix breaks it up considerably, making it relatively easy to wash down with the hydraulic guns and pump to washer, thereby increasing the matrix yardage; (3) the weight of this uplifted matrix on the undisturbed bed aids in breaking down the more solid matrix bank; and (4) the hydraulic mining operations are confined directly ahead of the pit car, which means fewer ditches to blast in the mining operations.

While this dragline and this practice in the removal of overburden are typical of those at other mines in the field, there are differences that should be mentioned. The size of draglines varies from 4-yd. to 9-yd. machines. Some draglines are supported on four sets of four-wheel trucks traveling on rails, instead of the caterpillar sets described above. Besides those of Bucyrus-Erie manufacture, there are also Marions, and recently a smaller dragline of the comparatively new Bucyrus-Monahan "walking" type. The last has distinct advantages over the other types with either caterpillar or truck support, in that it can be handled more easily in soft ground and with greater safety, and can

move away from any position in any direction without delay. However, draglines are expensive equipment and cannot be scrapped for new models that are more attractive.

The cost of removing overburden and dumping it into the mined-out cut alongside is low. During the 12 months of 1934 an 8-yd. Bucyrus-Erie dragline at one mine operated 4168 hr. out of a possible 8760 hr. and removed 2,471,781 cu. yd. at an average rate of 593 yd. per hour, consuming 0.555 kw.-hr. of electric energy per yard, at the following cost:

		COST PER CUBIC YARD ^a	
Labor:	Operating.....	\$0.0047	
	Moving.....	0.0003	
	Maintenance.....	0.0024	\$0.0074
Power:			0.0045
Supplies:	Operating.....	0.0002	
	Maintenance.....	0.0062	0.0064
Total cost.....			<u>\$0.0183</u>

^a Matrix weighs 125 lb. per cubic foot.

HYDRAULIC MINING

The retreating dragline, removing the overburden from the 210-ft. cut and dumping it into the mined-out cut alongside, is followed closely by the hydraulic mining of the matrix (Fig. 3). The distance from the face of the matrix bank to the face of the overburden bank varies between 200 and 400 feet.

Two hydraulic giants, or "guns," throw streams of water of about 2000 gal. per minute each (about 200-lb. pressure at nozzle), cutting and washing down the matrix bed (4 to 20 ft. thick) as well as the matrix that has been uplifted and set on top of the bed by the dragline. The hydraulic guns used are the "Climax," manufactured by Georgia Iron Works, equipped with 2-in.-diameter nozzles. They are of single-joint type wherein deflection of the stream of water is made both horizontally and vertically through a single joint, the gun having a horizontal range of 360° and a vertical range of 24° above and 24° below the center line. This is ample for all practical purposes. In operation each gun is anchored by bolting to foot blocks, usually of 6 by 8 timbers, 2 ft. long. The nozzle is manipulated by a 4 by 8 wooden pole, about 16 ft. long, which is attached by U-bolts to the barrel. This pole is tapered down at its end to convenient handle size.

The guns are fed by two 8-in. lines. A couple of hundred feet from the nozzles these unite in a 12-in. line from the hydraulic pumping station. The flow of water to the guns in the pit is controlled by 8-in. valves on the smaller lines, which are spiral welded galvanized iron. The 8-in. diameter has a No. 10 gage wall, while the 12-in. has a No. 8.

The joints are forged-steel flanges riveted on, and are packed with four-ply belt-stock drop gaskets of which the outside diameter is small enough to come within the circle of the bolts, which draw the joint up tight. The guns are moved up to within 25 ft. of the matrix face, a safe distance to avoid back splash that might injure the gunners. Mining proceeds usually in uneventful fashion. The 4000 gal. of water per minute washes down about 4 cu. yd. of matrix, the whole being pretty well worked up into a pulp. Through shallow ditches in the floor of the pit, the pulp flows to a sump or well near the guns.

Beside this well is the "pit car." The pit car is a much used unit in phosphate mining and consists of a Georgia Iron Works 12-in. single-stage sand pump directly connected to a General Electric 300-hp., variable-speed slip-ring, 2300-volt motor, assembled with the necessary electrical control, on top of a steel sled. Only the motor and its control is housed. To the suction end of the pump is coupled a 12-in. flexible suction set, which is dropped into the well and supported by a tripod and a $\frac{3}{4}$ -in. rope block and fall, so that the suction set may be raised and lowered. On the discharge side the pit-car pump is connected to the washer in the distance by a 12-in. pipe line. For this "rock line" 12-in. O.D. heavy-duty steel casing is used with peened-on cast-steel flanges at the joints. The joints are packed with four-ply belt-stock drop gaskets with outside diameter small enough to come within the circle of the bolts that draw the joint up tight.

As mining operations advance the rock line gets longer and it becomes necessary to place one, and later another, pumping unit in line. These are exactly like the pit-car unit but when so placed are called "lift" or "booster" pumps.

The efficiency of these single stage centrifugal pumps is, of course, low—about 50 per cent—but they are very simply constructed, easy to repair, cheap, and the most practical yet found for this kind of work. Many different types of centrifugal pumps have been tried out, including those equipped with internal water lubrication on the faces where sand and gravel cut deepest into the metal, but none has proved as trustworthy as the single pump now used.

As the face of the matrix recedes from the fixed position of the guns, the cutting effect of the streams diminishes. When the face of the matrix reaches a distance of 125 to 175 ft. from the guns, which happens about every 48 hr. of continuous operation, or when the slope of the floor of the pit or the ditches is so slight that the pulp does not easily flow to the well, it is then necessary to make a "move."

Mining operations are kept going while preparations for a move are being made. The electrical gang will have the pole line that carries power to the pit car advanced and ready to connect in. The floating gang will have the necessary additional pipe almost in place for both the hydraulic

line and the rock line. Drill holes for the new ditches and new well will have been drilled and loaded. Then, at a signal from the air whistle on the dragline, everything stops, and immediately thereafter the well is blasted (men at a safe distance), pipe lines are uncoupled, power lines to the pit car are cut, a 1¼-in. cable from a special drum under the floor of the dragline is attached to the pit car and the dragline operator draws it up to its location by the new well, the tripod is re-erected and the suction unit is installed. The dragline operator draws up the guns to their proper place, the extra pipes are rolled into place and bolted, and the pit-car power line is connected. The pit crew telephones the washer crew and the pumping station to be ready, because the pit is going to start pumping, and shortly mining of the matrix is again under way. The move takes from an hour to an hour and a half.

In matrix containing little or no clay, it is a question as to whether it would not be cheaper to excavate the matrix with the dragline, or with an additional dragline (operating solely for this purpose), the matrix being dumped into cars and hauled to the washer. This is being done today at one mine in the field. The cost of excavating the smaller yardage of matrix and spotting it in cars would probably be not more than 3¢ per cubic yard. With hydraulic mining and pumping costs at about 12¢ per yard, there is a margin here of about 9¢ per yard to cover transportation and subsequent pulping of the matrix. One additional 12-in. pump would be necessary at the washer in any event, to lift the pulp to the top of the washer. In very clayey matrices the cost of pulping might approximate the present cost of pumping.

The operation of the pit is in charge of a foreman under whose direction work seven men. Their duties are to operate the guns, blast the new ditches and well, repair leaks and blowouts in the pipe lines and attend to the shifting of the pit car and guns when a move is made.

The largest crew around a phosphate mine is the floating gang, usually about 20 men in charge of a foreman. Their job is to keep all equipment cleaned and in operating shape, including the pipe lines, run additional pipe lines, install or move booster pumps, repair roads, dams, ditches, etc. Their duties are practically all those not specifically allotted to somebody else, so therefore they go anywhere and everywhere they are needed, hence the name "floating" gang.

The electric gang, with five men under a foreman, is what its name implies. It is also a floating gang and is responsible for the running of power lines, telephone lines, the maintenance of all electrical equipment and circuits, not only on the mining operations but also on the washers, pumping stations and the drying plant.

Such an operation as has been described will deliver to the washer an average of 225 to 250 cu. yd. (in place) of matrix per hour. This, of course, varies considerably with the character of the matrix, whether it is

very clayey, and whether its clay is of the stiff compactly bedded variety, or soft clay. In pumping a soft clay matrix with 12-in. equipment, 275 cu. yd. per hour can be handled, compared to only 225 cu. yd. on a heavy sand matrix. When "sweeping" the pit just prior to a move, yardage will drop to as low as 150.

Phosphate matrix is abrasive material, so cast-iron pump shells last for about 350,000 cu. yd. The cast-iron runners or impellers will pump about 250,000 cu. yd. Heavy-duty steel casing, 12 in. outside diameter, lasts about a million yards—the wear being fast close to the flanged end. One man is usually kept busy at a mine re-peening steel flanges on a freshly cut end. Dents in the rock line soon wear through and are patched by welding. When the pipe wears too thin and splits open, or when pipe lengths have become too short, the scrap heap is ready for them.

The hydraulic lines of galvanized spiral welded pipe show little wear except that of corrosion or rusting.

The average cost of hydraulic mining and pumping to the washer at one mine during 1934 was 12¢ per cubic yard, as follows:

	PER CUBIC YARD	
Labor: Operating.....	\$0.0203	
Moving.....	0.0089	
Maintenance.....	0.0106	
Superintendence.....	0.0074	\$0.0472
Power:		0.0462
Supplies: Operating.....	0.0053	
Maintenance.....	0.0122	0.0175
Other direct costs:		
Moving, use of equipment.....	0.0008	
Maintenance, use of equipment.....	0.0002	
Upsetting matrix.....	0.0089	0.0099
		<u>\$0.1208</u>

During this year the pit was in operation 4413 hr. out of a possible 8760 hr., mining and pumping 1,024,999 cu. yd. of matrix an average distance to the washer of about 3000 ft. The average thickness of the matrix was 8.46 ft. There were consumed 5.7 kw.-hr. per cubic yard. When mining operations are near the washer, pumping costs are at the minimum, other things being equal. The difference in cost per cubic yard due to increased distance from the washer may be stated as an increase of 1¢ per yard for each booster pump in the line. The average distance each booster pumps is 1500 feet.

MILLING THE MATRIX AND DRYING THE PHOSPHATE

While it is not the purpose of this paper to describe in detail more than the mining practice in the phosphate field, some mention of the subsequent

processing of the phosphate to prepare it for market seems necessary for a better picture of the industry.

In the washer (Fig. 4) the incoming matrix, composed of sand, clay and phosphate varying in size from pebbles the size of one's thumb



FIG. 4.—WASHER OR RECOVERY PLANT.

The higher trestle shows incoming rock line from mining pit; the lower trestle supports tailings flume.

down to colloidal fineness, is processed by desliming, scrubbing, screening, classification and selective concentration in a modern ore-dressing plant treating 9000 long tons of matrix per 24 hr. The recovered phosphate represents from 85 to 90 per cent of that contained in the matrix. In one

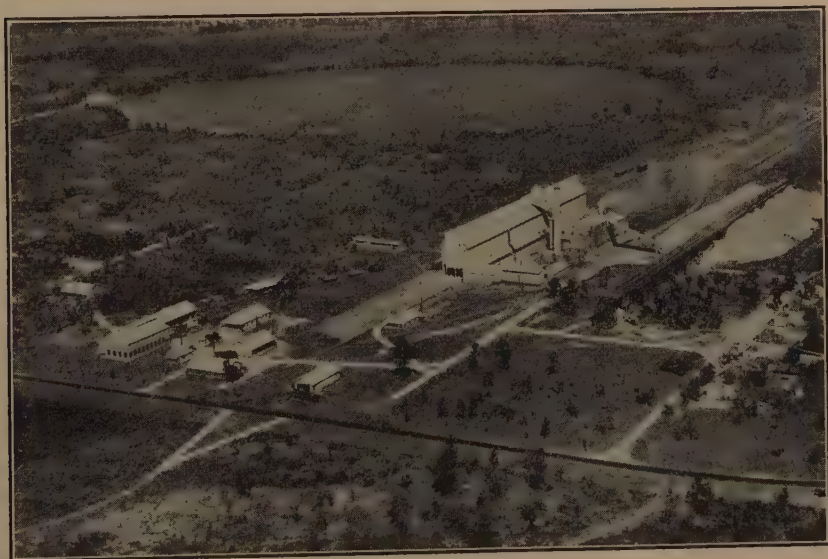


FIG. 5.—AIR VIEW OF MODERN DRYING PLANT.

Wet-rock storage trestle on right, mill building containing driers and kiln in center, dry-rock storage on left. Buildings in lower left foreground are shops, laboratories and storehouses. (*Aero-Graphic Corporation photo.*)

plant, more or less typical of the practice in the field, three differently sized products are separately and successively produced; viz., plus 5-mesh by screening, plus about 24-mesh by hindered-settling classification, and plus about 100-mesh by selective concentration. These three products

will vary slightly in grade from the same shift or same 24-hr. operation, and the production from day to day will vary in grade, as there is nothing selective in its mining.

The selection comes in the separation of the three products mentioned above, according to car analyses and their dumping into their proper place in the "wet-rock" storage pile at the drying plant. As it comes from the small bins at the washer, this wet rock contains from 10 to 25 per cent moisture, of which 4 or 5 per cent is absorbed within the pebble itself. This production moves daily over the Atlantic Coast Line R.R., the Seaboard Air Line Railway, or in a few cases over privately owned trackage in hopper-bottom open-top cars, to the central drying plant (Fig. 5).



FIG. 6.—CONCRETE TUNNEL AND BELT CONVEYOR UNDER THE WET-ROCK STORAGE TRESTLE.

The drying plant is of permanent construction, while the mining plants or washers are somewhat temporary in that they have a life of only 5 to 10 years in any one location. Often mining operations are as far as 20 miles rail haul from the drying plant. Around the drying plant usually are located the village and company office, the machine, electrical and carpenter shops, the storeroom and the laboratories. Drying plants consist of wet-rock storage, rotary drying and calcining equipment, and dry-rock storage. Wet-rock storage is always open storage, usually built up by dumping from a trestle, and usually recovered by a grab-bucket and sometimes recovered by a concrete tunnel under the storage pile (Fig. 6).

The drying equipment is a battery of oil-fired rotary driers, the largest being about 7 ft. 6 in. in diameter and 60 ft. long, having a capacity per unit of about 1200 tons of phosphate per 24 hr. A few of the plants in the field are equipped also with rotary kilns for calcining at from 1500°

to 2500° F. These are similar to cement kilns. One is 6 ft. diameter inside the firebrick lining, by 110 ft. long.

Dry-rock storage is covered storage, usually concrete and steel-bin construction, though there still are some wooden bins and one all-steel bin now under construction. The bins usually are built over one to four standard-gage railroad tracks, the phosphate loading by gravity into hopper-bottom decked-over cars (50 to 70 tons capacity) for shipment to port. For interior points box cars are used but some dunnage must be placed around the doors to prevent spillage.

All three parts of the drying plant are connected usually by belt-conveyor systems, making the whole operation highly mechanized. The phosphate usually is dried down to about 1 to 2 per cent moisture, which the buyer requires for good grinding qualities in his dry grinding plant.

DESTINATION

At Port Tampa (A.C.L.R.R.), Tampa and Boca Grande (S.A.L. Ry.) cargoes up to 9000 long tons are quickly loaded into ocean-going vessels for delivery to Atlantic and Gulf ports, as well as to Europe, Canada and Japan. At Port Tampa recently installed facilities now load phosphate cargoes at the rate of 1200 long tons per hour.

The commercial grades of phosphate pebble are:

GRADES	BASIS, B.P.L., PER CENT	ABSOLUTE MINIMUM, B.P.L., PER CENT
68 $\frac{5}{6}$ per cent.....	68	66
70 per cent.....	70	70
72 per cent.....	72	72
75 $\frac{1}{4}$ per cent.....	75	74
77 $\frac{1}{6}$ per cent.....	77	76

The I. & A. maximum is 4 per cent; the moisture maximum, 3 per cent. These grades are increased in number by sizing specifications and by low impurity specifications.

USES

About 85 per cent of the raw phosphate produced is acidulated with sulfuric acid to make "superphosphate," a necessary plant food in commercial fertilizers. Some 10 per cent or so is manufactured into di-sodium and tri-sodium phosphates, for silk weighting and for baking powder, the rest going into the manufacture of the many other phosphorus compounds used in industrial chemistry, photography, medicine, et cetera.

When Paris dressmakers require that milady's dresses shall "rustle," a few whites and many blacks in the Florida pebble field work a few more hours than when styles call for chiffons, for these require less weighting than taffeta, for which di-sodium phosphate is used. However, it is the farmer's dollar that moves most of the raw phosphates, for he must have them to grow his money crops and such crops must be grown if we are to eat, clothe ourselves, or reach for a cigarette.

DISCUSSION

(H. I. Smith presiding)

H. I. SMITH,* Washington, D. C.—In the selective recovery plant, what reagent, if any, other than water is used, and how much heat is required to reach the 2 per cent of moisture?

C. A. FULTON.—A fatty or resinous acid and petroleum are used. The rock is heated to about 225° F. for drying it to 2 per cent moisture.

H. E. CULVER,† Pullman, Wash.—Is the moisture much higher than 2 per cent as it is found in the ground?

C. A. FULTON.—Oh, yes, the sedimentary deposit is below the water level, the natural level of the water under the ground, so it is naturally wet. I suppose about 10, 15 or even 20 per cent.

G. R. MANSFIELD,‡ Washington, D. C.—Is the practice of using the dragline for removing overburden general now?

C. A. FULTON.—I know of no overburden being removed by hydraulic means; it is all done by dragline.

MEMBER.—What is the maximum size of the pebble?

C. A. FULTON.—Pebbles are usually not larger than the size of my thumb. There are clay balls and sometimes odd pieces of sand rock, or other stuff of that kind, as large as 4 in., but they are rare.

MEMBER.—What kind of lining is in the pumps and how long does it last?

C. A. FULTON.—The lining is cast iron. Other data are in the paper.

The typical pump is the single-stage 12-in. pump manufactured by the Georgia Iron Works. It is not as efficient as some other pumps, but for the kind of work we do we have found it more serviceable and more practical and more economical than the more efficient pumps.

G. S. RICE,§ Washington, D. C.—This paper by Mr. Fulton brings to mind what I think was the earliest use, for mining purposes, of the combined movable crane and dragline scraper. These had been used separately and in one or two instances together in constructing canals and levees but not in mining until 1907. In 1905, C. H. MacDowell, then president of the Armour Fertilizer Works, engaged me as a consulting engineer on certain phosphate properties and development in Tennessee and Florida, among them what later became the MacDowell mine in the Florida pebble phosphate field. Mr. MacDowell had been familiar with the previous developments in this field, which employed steam shovels and hydraulic methods, the hydraulic pressure being obtained by electric power-driven pumps. He thought that there was a possibility that there might be some cheaper method of excavating, especially if the production requirements were intermittent and also better methods of cleaning the pebble than then practiced might be found.

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† Head, Department of Geology, State College of Washington.

‡ Geologist, U. S. Geological Survey.

§ Chief Mining Engineer, U. S. Bureau of Mines.

In May 1907, after an extended study, I proposed the use of a movable crane with dragline scraper attachment, which up to that time, so far as I am aware, had been used in a limited way only for civil engineering projects—canals and levees. The proposed plan, which was later carried out, was to employ one or more combined movable cranes with dragline scrapers and run them on the top of the pebble phosphate, which was firm and practically level, the excavation being made in a straight line along one edge of the proposed pit. After the initial stripping and the first cut in the pebble phosphate had been done the plan was for the crane to dump the excavated waste overburden into the previously started pit and in a second operation or by another crane the underlying bed of pebble phosphate would be scooped up and dumped by the scraper bucket into small cars hauled by cable, running on tracks to the washery. Alternatively it was suggested that a belt or series of belts might be used to transport the pebble and matrix. The track or belt would be laid on the original surface and be moved laterally before each new cut was made. It was planned that the pebble and matrix would be dumped into a long ground bin, lined with concrete and that the material would be removed from this by small hydraulic guns washing into a sump, from which it would be raised to the elevated crusher and screens.

Following tests made at Chicago and at Joplin, it was recommended that the fine-sized material such as formerly went to the tailings be washed on jigs and tables, the results of the tests showing that the cost of this would be well repaid by the recovery from the washed fines. The first crane obtained was from the Lidgerwood Manufacturing Co., and it was the largest that had then been built, having a 70-ft. boom. The bucket had a capacity of $1\frac{1}{2}$ yd. The plant was constructed in 1908 and was, I understand, immediately successful.

H. S. MARTIN,* Mulberry, Florida (written discussion).—It seems to me that Mr. Fulton has commented too briefly on the recent development that has doubled the available phosphate reserves in this field. The successful treatment of this former waste material was first accomplished by the Phosphate Recovery Corporation. This company commenced commercial operations near Mulberry, Fla., in 1927, and the use of flotation concentration or tabling of the conditioned phosphate waste has since developed into a large industry. At the present time there are five large plants treating this former waste material by froth flotation processes and three plants treating this material by means of tabling after conditioning with flotation agents. Five different companies are operating plants that recover this finer rock. Two of these concentrating plants are now operating on dumps of old washer waste or "debris" accumulated some years ago. In the pebble district there are many old dumps containing several million tons of material that will eventually yield a large tonnage of phosphate concentrate. In at least two of these operations the yield of finished concentrate from the minus 14-mesh current washer waste is greater in tonnage than the original plus 14-mesh washer rock recovered in these plants.

Since the cost of removing overburden and the general overhead was formerly charged to 50 per cent of the present-day production, and since this type of flotation concentration can be done at low cost, the treatment of the fine washer waste has also materially lowered over-all costs where flotation concentration is used.

* Manager, Phosphate Recovery Corporation.

Drying and Processing of Pebble Phosphate in the Florida Field

BY CHARLES N. BECKER*

(New York Meeting, February, 1936)

THE practice of drying phosphate in Florida is as old as the industry, which began a little more than half a century ago. The methods, however, have changed considerably. At first, the natural process of exposure to the sun and air was used, later supplemented by wood fires beneath open-air stationary dry kilns and drying towers of various designs. It was not until the early nineties that rotary driers came into use, and at first they were of very small design and fired with wood. As the industry grew and tonnage increased, so likewise grew the number and size of rotary driers, until today the largest installations are 6 ft. up to 7 ft. 6 in. in diameter, and 55 to 60 ft. long, fired with crude oil.

The reasons for drying phosphate rock are: (1) to eliminate freight on water in the shipment and (2) to facilitate the good grinding qualities required by the buyers in their acidulation.

Phosphate pebbles vary materially in natural characteristics and each property has a direct bearing on the processing. In making comparisons of cost and operating conditions, the following are usually taken into consideration:

1. The hard dense pebbles, which contain very little interior moisture and 6 to 8 per cent total moisture, usually produced from the low-grade deposits.

2. The lighter, porous pebbles, which contain 4 to 5 per cent interior moisture and a total moisture ranging from 9 to 15 per cent.

3. The product of recovery operations using oily reagents, containing from 15 to 25 per cent moisture.

4. The size of the pebbles. Usually the plus 5-mesh material is considered as "coarse rock"; the minus 5-mesh plus 24-mesh is "fine rock," and the minute particles below 24 mesh are called "concentrate."

5. The amount of moisture allowed in the dried product. The practice is generally to dry down to 1.5 to 2 per cent moisture. There is, however, an actual range in guarantee of from 1 per cent maximum to 3 per cent maximum.

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FLOW DESCRIPTION OF ROCK MOVEMENT

The wet rock is received at the plant, usually on a schedule of twice a day, in open-top, bottom-discharge cars having a capacity of 18 to 50 tons each, and is placed on a siding to await the result of chemical analysis. After each car has been graded and marked for its proper place in the wet storage, it is weighed, run up an incline to an open bent trestle and dumped. From this point the rock is distributed to the mill either by steam-hoist recovery, by power hoes to elevators, or through an underground tunnel by belt conveyors to bins adjacent to the drying units, the method of transportation depending upon the size and location of the storage. From here the rock is fed to the driers by belts or elevators when the discharge of the bin is at floor level, or direct to table feeders or weight feeders when the bin discharge is above the driers.

There are two types of rotary driers in use at present—the small contraflow units, 30 ft. long with 4-ft. diameter, equipped with cast-iron spiral flights to lift and move the material forward, and the larger, 60 ft. long and inclined, parallel flow units, 6 to 7 ft. 6 in. in diameter, equipped with steel lifting flights. The material passing through the units is discharged either into a concrete pit and elevated to the dry-bin level or is discharged directly on “hot-rock” belts for conveying to the dry bins.

The dry-rock and calcined-rock storage is in the main a series of concrete bins with steel upper structure, covered and built above one or more standard-gage railroad tracks. These are either circular or straight sided of varying length, width and depth. The bins have overhead conveyor systems for distributing the material to its proper dry storage and cast-iron chutes and gates for discharge by gravity flow into cars placed directly below when it is ready for shipment.

The rock movement for calcining units is practically the same as the drying. However, as the temperature of the material discharged from the rotary kiln ranges from 1500° to 2500° F., it is necessary to reduce the heat to a point where the material can be handled by mechanical conveyors. This is accomplished by rotary coolers having lifting flights to spray the material through a contraflow of air, or by water-cooled rotary cylinders.

FACILITIES IN PLANT YARD

Trackage and motive power required for peak production and shipments, which often necessitate the handling of over a hundred cars a day, consist of two or more steam locomotives of the yard switcher type or the heavy-duty electric trolley system, and several miles of standard-gage railroad track laid out to accommodate the necessary separated storage requirements of the incoming empty dry-rock cars, the loaded

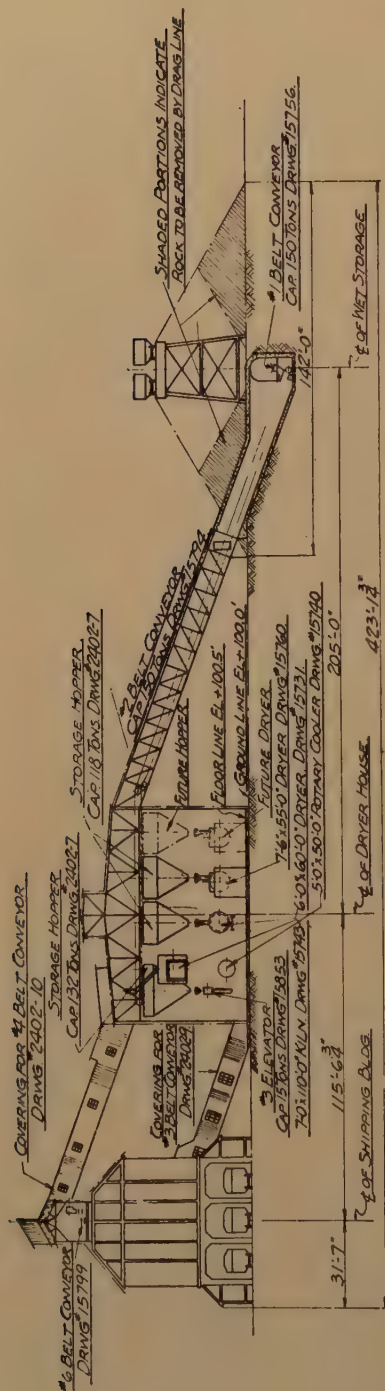


FIG. 1.—WET STORAGE, MILL BUILDING AND DRY STORAGE, WITH COVERED CONVEYOR CONNECTIONS.

wet-rock cars, the outgoing empty rock cars, the loading tracks for the dry storage, the scale track for weighing the cars and one or more incline tracks to the wet storage. One plant uses all 70-lb. rail, except on the incline and trestle, where 85-lb. rail is used, and has a total of 27,500 ft. of standard-gage track within the yard limits.

WET STORAGE

One wet-rock storage installation comprises 1170 ft. of double track, inclined dirt fill, on a 3 per cent grade to a level open bent-steel trestle 850 ft. long, 18 ft. 6 in. wide at top, and 35 ft. high. It is double-tracked with walkways along each side and in the center. Directly beneath the first 440 ft. of the trestle and below the ground level is a reinforced concrete tunnel, the interior of which is 9 ft. wide by 10 ft. high, straight sides with arched top and level floor. It has a series of 10-in. square cast-iron chutes and gates every 9 ft. 9 in. down the center and on each side of the arched top the entire length of the tunnel, to admit the rock by gravity from the storage above to steel hopper cars, which travel on a narrow-gage track astride the belt conveyor. This arrangement permits each unit of three chutes to discharge simultaneously into one hopper and where it is necessary to draw from more than one storage pile in order to mix rock on the belt conveyor, a second hopper is used.

The conveying system consists of a Goodyear "stacker" wet-material belt 24 in. wide, five ply, 32-oz. duck, with $\frac{3}{16}$ -in. rubber cover on the

carrying side and $\frac{1}{8}$ -in. rubber cover on the underside, and equipped with Metzler cushion edges and breaker fabric between plies. The belt runs on Link-Belt type 42, three-roll troughing antifriction carrying idlers, 5-in. dia. by 24 in. wide, spaced 4 ft. 2 in. apart, and 5-in. dia. type 40, single-roll straight antifriction return idlers spaced 8 ft. 1 in. apart, mounted on 15-in. stands bolted to the concrete floor. It is propelled by a General Electric, 10-hp., 220-volt, 1160-r.p.m. totally enclosed fan-cooled motor and a Link-Belt herringbone speed reducer to transmit 6.48 hp. with a ratio of 9.91 to 1, and low-speed shaft of 117 r.p.m., direct connected with flexible coupling and mounted on the same base with the motor. The head pulley is 30-in. dia. by 26-in. face, equipped with Dodge-Timken pillow block bearings and rubber belt scrapers, while the tail pulley is a horizontal gravity counterweighted take-up, 22-in. dia. by 26-in. face, with similar bearings. The belt, traveling 200 ft. per minute, conveys 150 tons per hour and discharges into a stationary steel hopper at the head end of the tunnel and on to a second belt running at right angles 300 ft. per minute out of the tunnel, up a 19° inclined covered way to the top of the mill building (Fig. 2). The belt and conveying equipment are the same as in the tunnel system.

Because of seasonal heavy rainfalls and the drainage water from the rock, the tunnel is provided with terra cotta pipe drainage at the base of the outside walls, through which excess water flows into a steel tank inside the tunnel and from there is pumped away by an automatically controlled Allis-Chalmers $2\frac{1}{2}$ -in. suction and discharge water pump, direct connected to a 5-hp., 220-volt, 3500-r.p.m. Allis motor. The seepage within the tunnel drains to a sump at the head end and is picked up either by steam siphons or by a high-speed water pump. The tunnel is well lighted, having 90 lamps of 40 watts each, with metal guards, located between the rock chutes on the side walls. Ventilation is provided by a suction fan at the north end, which draws fresh air the entire length of the tunnel and exhausts up a 10-in. pipe through the top of the tunnel and rock pile into the open.

The storage of wet rock above the 440-ft. tunnel, when filled, will permit a gravity flow to the tunnel of approximately 30,000 tons. It is divided into 10 piles or grades. As a pile becomes filled it is set out beyond the trestle dumping area by an oil-fired, steam-operated industrial Brown-hoist with 55-ft. boom and a 2-yd. grab bucket. Likewise, as the storage directly above the tunnel becomes empty, and the incoming wet-rock cars are not of the grade being dried, it is necessary to reclaim from the piles set out on either side of the trestle. With these facilities, an inventory of 80,000 tons can be stored and reclaimed direct to the tunnel. The rock stored in the 410 ft. beyond the tunnel length of the trestle must be picked up by the hoist and put into wet-rock cars for dumping over the tunnel area. With this additional space a total wet-rock inventory of 150,000 tons is possible.

THE MILL

The latest plant to be erected in the field is an all-steel structure 140 ft. long by 80 ft. wide, 52 ft. high for approximately half its length and 40 ft. 6 in. high over the remainder, which houses all the drying and calcining equipment (Fig. 2). The siding, roofing, flashing and ventilators are of Robertson protected metal, painted aluminum inside and out. Numerous tilting windows in steel sash on all sides of the building provide ample light and air.

The rock conveyed from the tunnel enters the mill near the top and about the center of the building, so that it may be deposited by an electric traveling hopper into any one of the three 80-ton straight sided, tapered-bottom steel wet bins with shut-off gates, which are arranged side by side below the conveyor and above the drying and calcining units.

The gravity flow from the bins to each drier is regulated by an Allis-Chalmers 36-in. revolving table feeder, powered with a G.E. 3-hp., 220-volt, 850-r.p.m. motor direct connected to a Foote Bros. speed reducer, type H-6-S, ratio 23 to 1, 40 r.p.m.

There are two rotary driers of the largest size in this building, one a single-shell Ruggles-Coles incline drier, 7 ft. 6 in. in diameter by 55 ft. long, with eight rows of interior cast-iron spiral flights bolted to the first 6 ft. of the feed end to move the wet material forward and 12 rows of steel straight-line lifting flights over the remaining length of the cylinder. The drive consists of a G.E. 60-hp., 2200-volt, 900-r.p.m. slip-ring ball-bearing motor and a 9-belt Texrope drive having a 2 to 1 reduction to a heavy-duty Cleveland two-gear, oil-immersed, enclosed 20 to 1 ratio speed reducer and driving pinion, which is in mesh with an externally driven gear attached to the drier shell, which rotates the cylinder 12 times per minute. The drier is set on a pitch of $\frac{3}{4}$ in. per horizontal foot and has a forward and rear riding ring with an 8 ft. 9-in. dia. by $7\frac{1}{2}$ -in. face and two pairs of cast-iron trunnions, each 25-in. dia. by 8-in. face, in rocker arms supported by a cast-iron base mounted on heavy reinforced concrete pillars.

The furnace, which rests on a raised concrete platform at the feed end of the drier, is a steel box braced with 6-in. channel iron and is 12 ft. long by 7 ft. 6 in. wide by 8 ft. 6 in. high. It is lined throughout with 13 in. of A. P. Green Empire firebrick, has a high arched top and in the back is graduated down in funnel shape to a circle 7 ft. 6 in. in diameter, where it connects with the rotating seal ring attached to the drier. On the front of the furnace are the burners, fuel supply, return lines, meters and excess-air portholes.

Connected to the discharge end of the drier by a seal ring is the draft flue and 36-in. Clarage fan, 690 r.p.m., with a four-belt Texrope drive and G.E. 30-hp., 220-volt, 1160-r.p.m. ball-bearing motor, mounted on a steel platform above the drier. The fan discharges into a cyclone 25 ft.

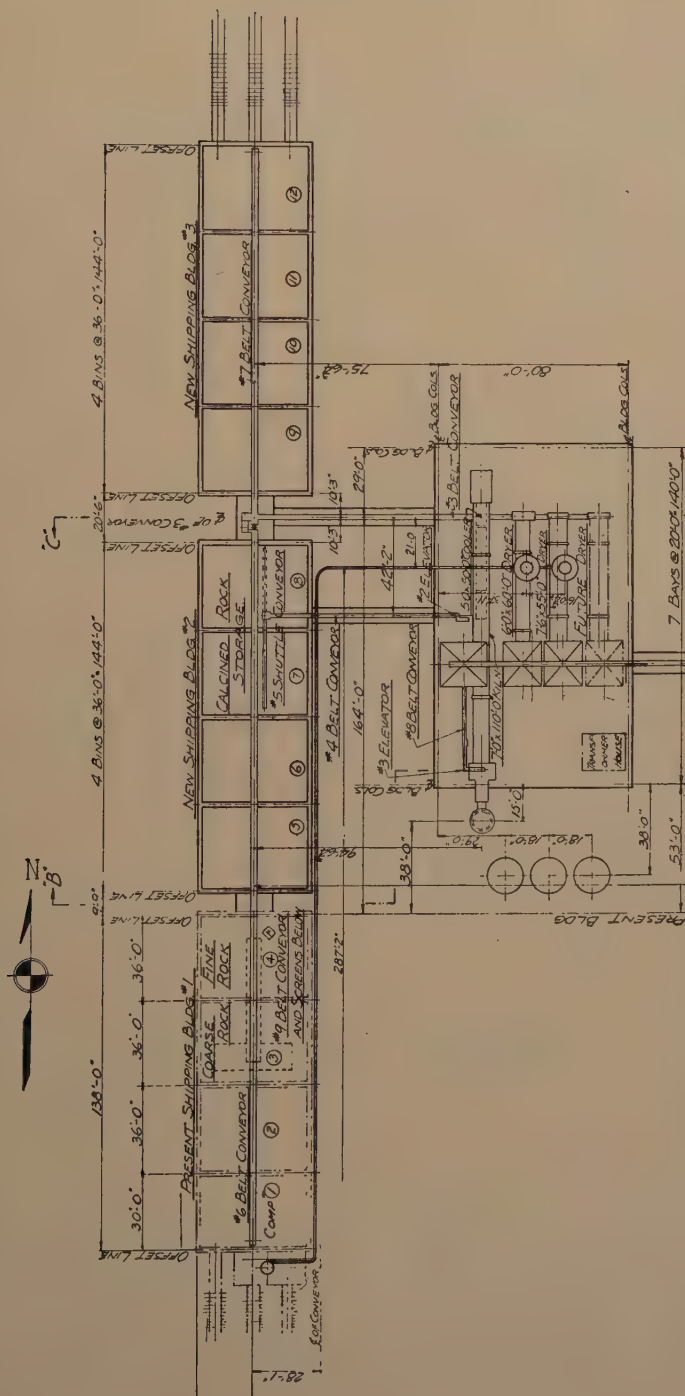


FIG. 2.—ARRANGEMENT OF EQUIPMENT IN MILL AND IN DRY-STORAGE BUILDING. NOTE PROVISION FOR FUTURE DRYING UNIT.

high by 12 ft. in diameter, with 6-ft. stack and bottom-tapered to 6-in. diameter.

The second unit is a single-shell incline Allis-Chalmers drier, 6-ft. dia. by 60 ft. long, mounted beside the Ruggles-Coles drier and having approximately the same supporting and auxiliary equipment, except that the reduction in speed between the 60-hp., 1200-r.p.m., 2200-volt, G.E. slip-ring motor and the driven gear attached to the shell is accomplished through a series of open gears and pinions down to 12 r.p.m. on the cylinder.

The Bethlehem-Dahl system of crude-oil firing is used for the driers. This consists of two stationary burners for each unit, with atomizers and tips capable of burning 80 gal. of bunker C oil per hour each. The essentials necessary for proper performance of this type of burner are superheated oil at high pressure to the burner tips and sufficient draft through the air veins surrounding the regulating pot to produce a swirling motion, to atomize the oil. This is accomplished by a small, oil-fired stationary boiler providing steam to the superheaters, which raise the temperature of the oil to 260° F., and a 2-in. suction and discharge Kinney rotary oil pump, type SD-422, gear-driven by a 5-hp., 220-volt, 1150-r.p.m., G.E. motor, which raises and maintains a constant 200-lb. pressure in the lines up to the burners. Oil is furnished to the pump by gravity from three steel storage tanks outside the mill building, having a total capacity of 80,000 gal. The necessary air for the burners is furnished by the Clarage fan, at the opposite end of the cylinder to provide parallel flow, which draws the air through the veins of the burners and the port holes. This fan is rated 30,000 cu. ft. per min., 4½-in. static pressure, 42 boiler hp. at 400° F., and provides 0.60 in. of draft, atmospheric temperature at the burners. The result is complete combustion of 160 gal. of oil per hour within a furnace having an area of 765 cu. ft. and a heat liberation of 31,000 B.t.u. per cubic foot, which does not contaminate the rock pebbles with particles of oil.

During a recent 110-hr. test, using both driers and rock of all the various sizes and moisture contents, there was dried 10,500 tons with an

Labor: Operating.....	\$0.019
Maintenance.....	0.009
Superintendence.....	0.005
Total labor.....	\$0.033
Power.....	0.014
Fuel oil.....	0.105
Supplies: Other operating.....	0.004
Maintenance.....	0.033
Total supplies.....	0.037
Total direct operating cost.....	<u>\$0.189</u>

average ingoing moisture of 13.8 per cent and an average outgoing moisture of 1.3 per cent, a difference of 12.5 per cent, or 26,880 lb. of water evaporated per hour. There was consumed 37,000 gal. of oil, or 337 gal. per hour—an average of 3.5 gal. per ton dried. For the year 1934 the cost of drying rock was 19¢ per ton, as shown on page 288.

Of the 365 days in the year, or 17,520 possible drier hours, these two units ran 5385 drier hours, or 31 per cent of the total time. Three crews of six men each, working 8-hr. shifts, and one foreman, are required to operate the mill on a 24-hr. day basis; a crew consisting of a fireman, oiler, tunnel tender, feed tender, dry-bin tender and sweeper.

ELECTRICAL EQUIPMENT

Three-phase, 60-cycle current at 2300 volts is delivered to a central brick-enclosed switchroom containing all the distribution transformers, switches and fuses. Three 75-kva. transformers supply the 220 volts for all the low-voltage motors through individual disconnecting switches and circuits. All motor operations are controlled by magnetic switches with push buttons, arranged on a raised concrete platform just outside the switchroom at the front of the mill building. To obtain proper operation of the various units and to avoid choking elevators or conveyors with phosphate rock through mechanical failure of any unit, all circuits are electrically interlocked and must be started in the proper order. A system of double indicating lights, mounted on a panel above the control platform, has been provided to check closely all the motors and their operation. Complete or partial failure of any unit can be detected at a glance and repairs started with a minimum of delay.

Electrically operated solenoid valves interlocked with the draft-fan motors are provided for each furnace to avoid the forming of an explosive mixture through failure of the proper operation of the fans. The 30 motors in the mill, tunnel, and dry-storage building consume 285 kw. per hour when all are in operation, and have a total rating of 440 hp., of which the 2200-volt motors have a total rating of 200 hp., 51.3 amp. and pull 43.0-amp. load, while the 220-volt motors have a total rating of 240 hp., 645 amp., and pull 555-amp. load.

DRY STORAGE

Rock discharges from the driers at a temperature of 212° F. to 280° F., down a cast-iron chute through the bottom of the draft flue, on to a Main Belting Co. ABC seven-ply hot-rock belt, 24 in. wide by 320 ft. long. This belt runs at a speed of 200 ft. per min. on Link-Belt anti-friction troughing conveyor with 30-in. head pulley and 22-in. tail pulley, driven by a G.E. 10-hp., 220-volt, 1200-r.p.m. totally enclosed motor

through a Foote Bros. speed reducer. It runs through a concrete pit below the floor level, past the end of both driers and up a covered incline out of the mill to the dry-storage building.

Midway of the conveyor is a Merrick weightometer, which records the weight of the material passing over it. Automatic sampling of the dried rock is accomplished with a General Engineering Co. sampler, operated by a G.E. time switch and $\frac{1}{4}$ -hp., 220-volt, 1725-r.p.m. motor, which completely cuts the stream of rock four times an hour as it leaves the belt for the elevator, 2 qt. of material being taken with each cut, and quartered in a four-way splitter. One part is piped to a sample bin and three parts dropped to the elevator pit.

From this point the rock is lifted to the top of the dry-storage building by means of a Link-Belt single-bar roller chain elevator 55 ft. high, with 24 by 8 by 12-in. type M.F. buckets spaced every 12 in., and head and tail sprockets with Dodge-Timken roller bearings. It is operated at a speed of 125 ft. per min. with a G.E. 20-hp., 220-volt, 900-r.p.m. motor through a Link-Belt herringbone speed reducer at a ratio of 15 to 1.

Two hot-rock belt-conveyor systems of the same specifications as the one previously described, but of different length, and running in the opposite direction to each other over the top of the dry bins, distribute the material to its proper storage through electrically operated traveling hoppers.

Provision has been made for selective sizing of the dry rock by mounting between two of the bins and below the conveyor belt two Link-Belt single-deck 4 by 8-ft. vibrating screens set side by side. The rock held on the screens drops into one bin, while the rock passing through the screens is discharged on a short belt for conveying to the other bin.

The dry-storage building is a reinforced concrete structure in three sections, each divided into four compartments, or bins. The bottoms of these bins are 20 ft. above the ground level and are supported by large reinforced concrete pillars spaced 6 ft. apart lengthwise of the building and 15 ft. apart over the width. Each bin is 36 ft. long, 45 ft. wide and 28 ft. high, and holds 2000 long tons of dry rock. The roof and monitor sections are all steel with Robertson protective metal covering.

Over-all dimensions of the building containing the 12 dry bins, with a total storage capacity of 24,000 long tons, are: 426 ft. long by 45 ft. wide by 80 ft. high to the top of the monitor.

Safety measures for the protection of the workmen are given much attention by enclosing drives and elevators, and in the use of arrowhead steel grating and handrails on all walkways and stairs.

Gravity-flow loading from any bin to cars on all three tracks is by numerous 10-in. square cast-iron chutes through the floor, having shut-off gates and 12-ft. long flexible steel spouts that can be moved quickly from

door to door in the top-hatch cars. The average cost of loading dry rock into cars is approximately $\frac{3}{4}$ ¢ per ton for labor.

A hand sample with scoop is taken from each car loaded and is delivered to the laboratory for chemical analysis, as a control in the daily all-rail domestic shipments and for the loading sequence of the cars at port, in order that a homogeneous cargo may be supplied.

THE DUST

Within the cylinders during the drying process there is an explosion of the wet pebbles as they first contact the intense heat, also a scrubbing action caused by the forward progress of the rock, which produces 2 or 3 per cent of very fine particles termed "dust." These particles are carried to the cyclone by the draft fan in parallel-flow driers. A typical screen test is as follows:

RETAINED ON	PASSING	PER CENT
50 mesh	40	0.42
80 mesh	50	5.58
100 mesh	80	7.70
200 mesh	100	20.56
	200	65.74

Total 100.00

The function of the cyclone is to separate the particles from the gas, the dust falling to the bottom and the gas escaping up the stack. By means of a butterfly valve at the base of each cyclone, the dust is evenly distributed to the conveyor pipe line that carries it out of the mill building to a completely enclosed concrete bin at the front end of the dry-rock storage building. Conveying of the dust is accomplished with a Fuller Company No. 2 Airveyor tube filter with enclosed cast-iron rotary discharge gate connected to a 14 by 27-in. positive-pressure rotary exhaustor with muffler. This is driven by a G.E. 40-hp., 2200-volt, 1750-r.p.m., squirrel-cage induction motor.

CALCINATION

Calcination is the heating of phosphate rock to temperatures of 1500° to 2500° F. for a period of a few minutes to an hour or longer. This is done to meet the requirements of acidulators, principally manufacturers of di-sodium and tri-sodium phosphates, who desire a phosphate rock practically free from organic matter. The 1 to 3 per cent of organic matter in uncalcined rock prevents production of water-white phosphoric acid, necessary in making water-white sodium phosphates.

The unit in use at one plant consists of an Allis-Chalmers rotary kiln, 7 ft. outside diameter by 110 ft. long, mounted on two large reinforced-concrete foundations approximately 19 ft. above the floor. It is lined throughout with Empire rotary-kiln blocks, 6 by 9 by 4 in., 72 by 84-in. dia. This operation requires contraflow, hence draft is provided on the

wet end by a tapering, reinforced-concrete chimney with radial brick lining, full height 113 ft. by 7 ft. in diameter. The heat is produced by a similar method to that described in the drying but with the furnace on the discharge end of the cylinder. The drive is a series of heavy reduction gears and pinions powered by a G.E. 40-hp., 2200-volt, 1200-r.p.m. slip-ring motor connected to the main drive shaft by a six-belt Texrope drive which revolves the kiln on riding rings and heavy trunnions once every 82 sec. The unit is set on an incline of $\frac{3}{4}$ in. per horizontal foot to produce a forward movement of the rock. At the speed the cylinder is turning it requires one hour for rock to traverse the entire length of the kiln. In order that the bed of rock may be always the same, a Hardinge constant-weight feeder is used to measure the flow from wet bin to conveyor for elevating to the top of the entrance chute.

To facilitate mechanical conveying and weighing of the hot material from the mill to storage bins, a 5 by 50-ft. inclined rotary cooler with lifting flights is set directly below the kiln at the discharge end. This is operated by an open gear unit and 20-hp., 220-volt motor. The reduction of temperature to 300° F. is accomplished by drawing a contraflow of air at atmospheric temperature through the cylinder with a No. 33 Clarage fan, rated 17,000 cu. ft. per min., $4\frac{1}{2}$ static pressure, 798 r.p.m., 23 boiler hp. at 600° F., powered by a G.E. 20-hp., 220-volt motor. Exhaust of the fan is through a cyclone for dust separation and the stack gas at 600° F. is piped to the firebox to utilize its heat in the requirements of excess air for the burners, the remainder being by-passed into the open.

A Leeds & Northrup electric recorder and thermocouple register the temperature of all stack gases and rock discharges every 15 min. of operation, and the kiln discharge is checked by optical pyrometer hourly.

Costs of calcination vary according to the temperature attained. The following cost and operating statistics schedule is based on the average of the year 1934 for temperatures not exceeding 1600° F.:

Labor: Operating.....	\$0.171	
Maintenance.....	0.028	
Superintendence.....	0.021	
Total labor.....		\$0.220
Power.....		0.057
Fuel oil.....		0.556
Supplies: Other operating.....	0.003	
Maintenance.....	0.061	
Total supplies.....		0.064
Total direct operating cost.....		\$0.897
Work days.....	365	Kiln hours operated..... 5952
Possible hours.....	8760	Percentage of hours..... 68
Tons per hour.....		9

DISCUSSION

(*W. H. Coghill presiding*)

W. H. COGHILL,* Rolla, Mo.—What is the reduction of moisture in the drying plant?

C. A. FULTON,† Baltimore, Md.—Phosphate rock comes into the drying plant with from 6 to 25 per cent moisture. We endeavor to dry down to about 1.5 per cent. Most phosphate rock contracts have an outside limit of 3 per cent moisture, but the trade has become accustomed to lower moisture and expects the rock around 1.5 or 2 per cent.

MEMBER.—You speak about blending the different parts of the product. I suppose that the product is sold in different grades.

C. A. FULTON.—Yes. The commercial grades of phosphate rock are, roughly, 68 per cent B.P.L., another 70 per cent, another 72 per cent, and then 75 and 77 per cent, which is about the highest grade produced in the pebble district. The grade is the bone phosphate of lime content (tri-calcium phosphate), a relic of the days when bones were acidulated to produce phosphates or available phosphoric acid for plant food.

The mining is not selective except in a very broad sense, in that the mining operations are carried on in a high-grade or medium-grade or low-grade field. As the cars come into the drying plant in the morning they have been sampled at the producing plant, and then, somewhat later, when their grade is known, the cars are dumped according to grade. So often the producer runs short in one grade—say in the 72—and by mixing 75, say, with 70, he can produce just the grade he wants. Very many customers like to buy a 70 per cent grade, for instance and do not like to pay for units over 71 per cent because many of them have small acidulating plants where their chemical process is one of rule of thumb and not one of exact measurements. They are accustomed to treat a certain grade and they do not like too high a grade because, lacking chemical control, they do not put enough acid with it to get the benefit of the units of B.P.L. over, say, 71.

MEMBER.—What are the advantages of feeding the driers at the hot end rather than at the cold end?

C. A. FULTON.—The fine rock does not cake up in the front of the drier. With the old driers that are still used in the district where the rock is fed at the cold end it is difficult to dry fine concentrates. By fine, I mean a rock finer than 35-mesh. There is a tendency for the wet rock to build up around the cooler end of the drier.

We have used both methods. In a new plant we put in driers in which the rock flows in the same direction as the gases.

MEMBER.—Is there any difficulty with the dust becoming wet or damp, and does that affect the operation of the pump for handling the dry dust?

C. A. FULTON.—We have no difficulty with the dust becoming wet.

H. HANSON, Toronto, Ont.—As I understand it, the hot calcine goes to the cooler before being conveyed to the bins. It would seem to me that it would be more difficult to cool than to heat because of the degree of temperature.

* Metallurgist, U. S. Bureau of Mines.

† President, Southern Phosphate Corporation.

C. A. FULTON.—The rock as discharged from the kiln is anywhere from 1500° to 1700° F. It falls to another cylindrical steel tube equipped with flights, which lift the rock up and let it fall down through the cool air stream. I do not recall the exact dimensions, but I should say it is about 30 ft. long and 5 ft. in diameter. The air is drawn through this cylinder in sufficient volume to cool the calcined rock as it falls.

The cooling air goes to a suction fan, and then to a cyclone, which removes the dust. The calcining and cooling operation produces a great deal of dust. After the dust is removed, the air is returned into the firebox of the kiln, and the heat conserved. It is considerable heat, as may be judged by the fact that in the beginning we did not have the installation entirely completed, and we operated without returning the air to the kiln. There was a 50 per cent increase in fuel consumption as compared with returning the hot air to the kiln fire box.

MEMBER.—Does steam or dust, or both, come out of the stack?

C. A. FULTON.—Steam, but not entirely free from dust.

Ore-dressing Practice with Florida Pebble Phosphates, Southern Phosphate Corporation

By J. W. PAMPLIN,* JUNIOR MEMBER, A.I.M.E.

(New York Meeting, February, 1938)

SOME 40 miles east of Tampa is the center of the Florida pebble phosphate deposits. These are of Pliocene age and consist of several members of the Bone Valley formation.¹

Physically the phosphate-bearing formation, locally called the "matrix," varies in thickness from nothing to about 25 ft., usually averages 10 ft., and consists mainly of clay, quartz and phosphate pebble, the latter particles varying in size from colloidal to one inch diameter. The pebbles range in color from dark gray, through white, to cream, and in hardness from that which must be broken with a hammer to that which can be scratched with the fingernail. They assay from 66 to 80 per cent bone phosphate of lime (called "B.P.L." in the trade) or tricalcium phosphate ($\text{Ca}_3\text{P}_2\text{O}_8$) and are secondary apatites. They form from 15 to 50 per cent by weight of the matrix. Over the matrix lies a varying thickness (5 to 40 ft.) of overburden composed of quartz and clay.

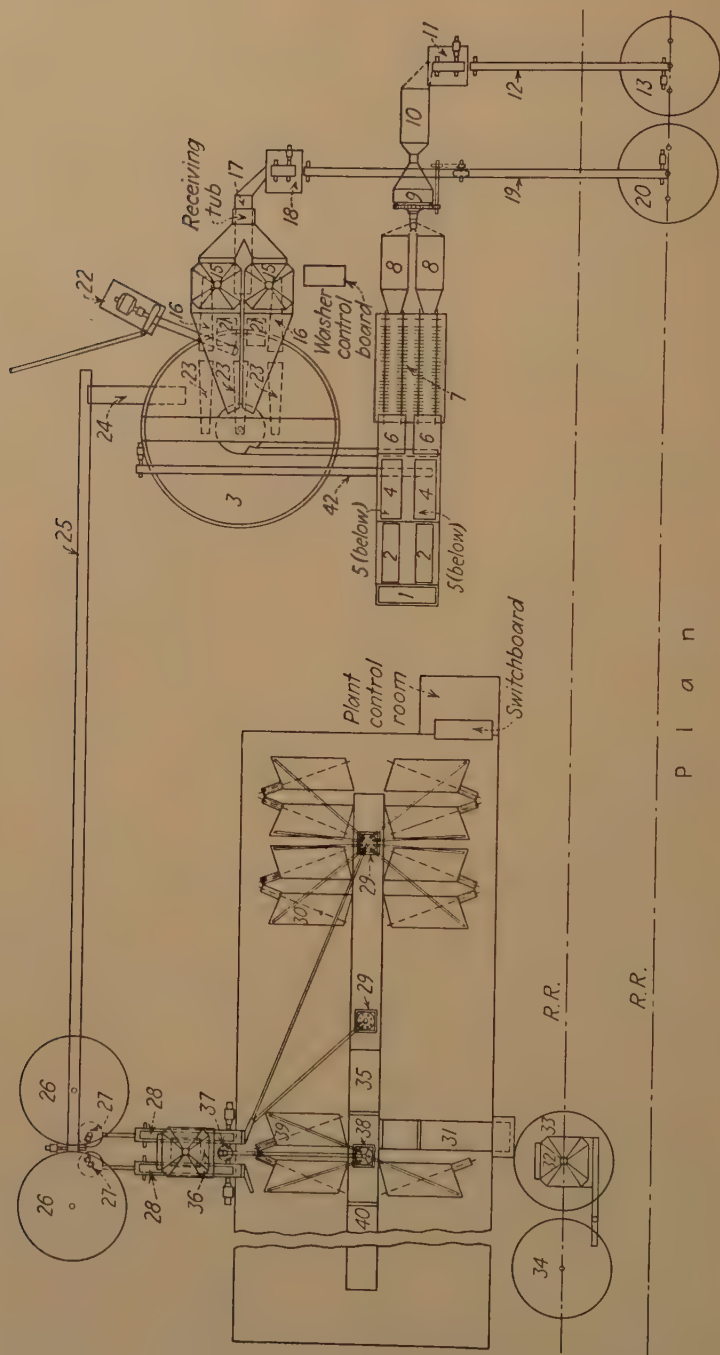
Briefly, mining consists of two operations² in open pits about 200 ft. wide by $\frac{1}{2}$ mile or more in length. In the first step overburden is removed by large electric draglines at the rate of about 700 cu. yd. per hour. This is followed closely by hydraulic mining of the matrix at the rate of about 300 cu. yd. per hour. This matrix yardage is moved from an improvised sump in the floor of the pit to an ore-dressing plant, sometimes a mile distant, by one, two or three "pit-cars" (pumping units) working on a 12-in. pipe line called the "rock line" to distinguish it from the high-pressure water line feeding the hydraulic giants, which is known as the "hydraulic line."

This 700 yd. of overburden and 300 yd. of matrix—1000 yd. in all—amounts to about 1500 tons of material moved each hour. The overburden is dumped into the mined-out pit alongside the pit being mined. The matrix moves to the ore-dressing plant at the rate of about $7\frac{1}{2}$ long tons in 4000 gal. of water per minute—a ratio of approximately one to two by weight, or one to six by volume. This constitutes the "mill heads," but in the pebble district it is called the "matrix."

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* Metallurgist, Southern Phosphate Corporation, Bartow, Florida.

¹ References are at the end of the paper.



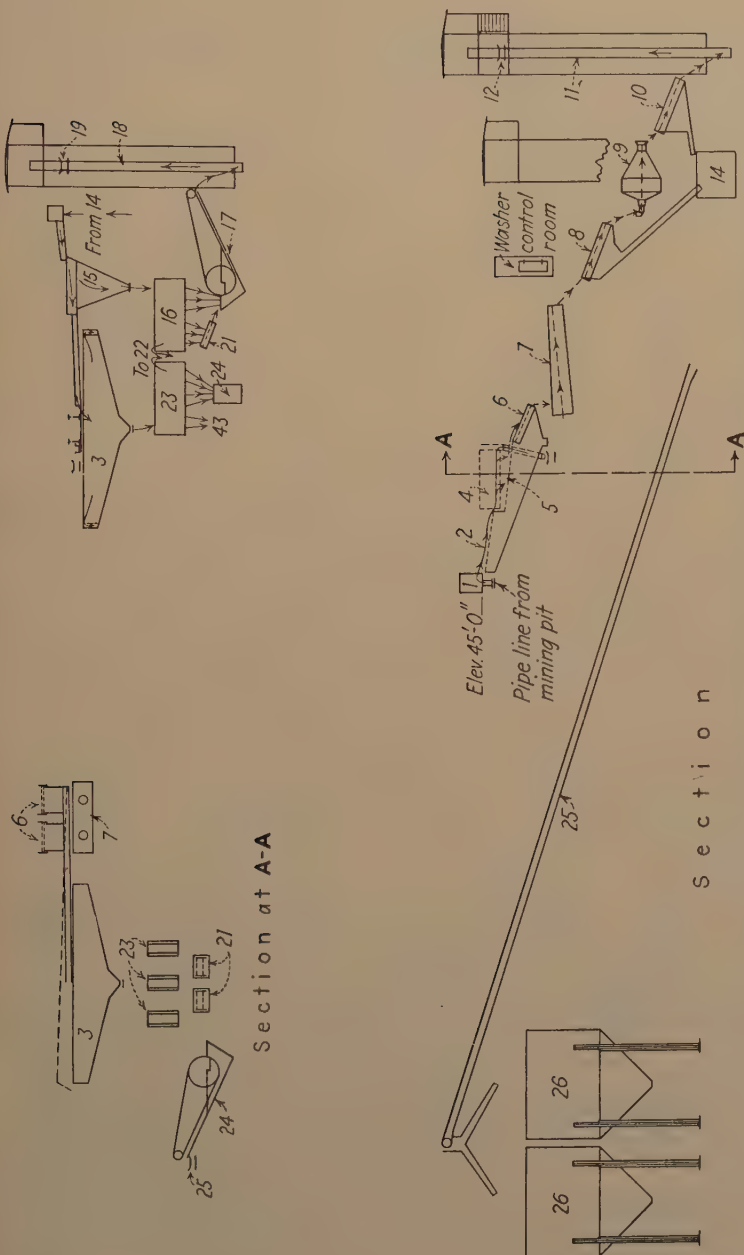


FIG. 1.—GENERAL LAYOUT OF SAN GULLY WASHER AND RECOVERY PLANT.

The ore-dressing plant might be called the "mill," but instead that part handling two-thirds of the bulk has inherited the name "washer," and the remainder is called a "recovery plant." Actually, all of it is a recovery plant, the "washing" of the phosphate being a necessary but incidental step in the winning of the phosphate pebble from the matrix. For the purpose of this paper this entire ore-dressing plant will be called a "washer."

To the phosphate company a washer, even though costing about \$200,000, is a temporary plant. It is located on an acreage of practically level phosphate land, so that the matrix within a mile may be mined to it. Further than a mile away from the washer, "water transportation" of the matrix by means of pumps and piping becomes costly and difficult and it is cheaper to dismantle the washer and re-erect it on an adjacent location. However, during its life of 8 to 10 years in a given location it may have produced as much as two million tons of phosphate, and with the more recent practice of erecting steel (instead of wooden) structures, probably only one-half of the above cost will have to be written off against the two million tons—about five cents per ton.

Each of the seven operating companies in the Florida pebble field, producing annually between two million and three million long tons of phosphate, requires from one to three such mining locations (pit and washer) as part of its current setup, because each mining location may produce only within narrow limits such as high-grade (74 to 80 per cent B.P.L.), medium (70 to 75 per cent B.P.L.), and low-grade (66 to 70 per cent B.P.L.). Each company has a permanent drying plant with large wet and dry rock storage facilities to which their mining locations feed wet phosphate pebble, known in the trade as "phosphate rock" when dried to about 2 per cent moisture.³

The problem confronting the ore dresser in the pebble phosphate field is first to rid the phosphate pebble of all clay, then to separate the finer phosphate pebble from quartz of the same sizes.

The first operation is partly accomplished in the rock line as the matrix comes to the washer. It is further accomplished by trommels, which remove the large clay balls; by log washers, which break up the smaller clay balls; and by Hardinge mills, which slime the fine clay balls. Dorr deslimers and Allen tanks remove the slimed clay and the very fine sands to waste. A typical analysis of the matrix when all clay is slimed is shown in Table 1.

The general layout of the washer and recovery plant is shown in Fig. 1. Different views are shown in Figs. 2 and 3, and the flowsheet in Fig. 4.

The plus 14 mesh and the minus 14 plus 20-mesh fractions are clean phosphate pebble. The former is of lower grade because of its higher silica content, but in either case this silica is a part of the pebble. In the very coarse sizes, plus $\frac{1}{2}$ in., it may often be seen as veinlets, while in the

finer sizes it is microscopically invisible. Below 20 mesh, quartz grains commence to "dilute" the phosphate pebble, in increasing amounts as the size decreases. With the clay in suspension as slime, it is easy to separate

TABLE 1.—*Typical Analysis of Matrix after Clay Is Slimed*

Mesh	Weight Per Cent	Bone Phos- phate of Lime, Per Cent ^a	Silica and Insoluble Silicates, Per Cent ^a	Distribution of B.P.L., Per Cent
Sands				
+14.....	13.1	74.80	9.15	31.8
-14 +20.....	3.7	76.00	7.64	9.1
-20 +28.....	2.2	60.32	24.14	4.3
-28 +35.....	5.3	44.51	45.16	7.7
-35 +48.....	7.8	32.03	60.20	8.1
-48 +65.....	9.4	21.27	72.52	6.5
-65 +100.....	19.6	10.74	86.59	6.8
-100.....	16.1	9.90	87.30	5.2
Slime.....	22.8	27.83	48.66	20.5
Total and average.....	100.0	30.84	58.04	100.0

^a The balance is principally lime in excess of B.P.L. ratio, iron and aluminum oxides, organic matter and water of combination, and fluorine.

the plus 5-mesh pebble by screening, and the plus 20-mesh (and sometimes finer) by hindered-settling classification.⁴

The second problem then is to separate the phosphate pebble from the free quartz. This is done by selective concentration, for the difference



FIG. 2.—EAST VIEW OF SAN GULLY WASHER.

in specific gravity between the two is so slight that ordinary gravity concentrating methods cannot be used.

Many of the plants in the field use flotation cells, with a reagent treatment that includes bringing the pH of the feed to the recovery plant from around 7 up to 8.0 or 8.5; a reaction on the surface of the pebble between the lime in the pebble and a fatty or resinous acid (or both); then a filming of the pebble with petroleum oil; and the addition of a frother to aid in the levitation of the phosphate pebble. However, for such a treatment the plus 35-mesh pebble must be ground if flotation tailings are to be kept below 10 per cent B.P.L. In most of the high-grade phosphate pebble deposits the pebble is extremely soft and such pre-flotation grinding results in much phosphate sliming. As most of the minus 100-mesh fractions (including such phosphate slime) are classified



FIG. 3.—NORTH VIEW OF SAN GULLY WASHER AND RECOVERY PLANT.

out prior to the flotation treatment, this adaptation of an excellent method for fine sizes, to the treatment of coarse sizes, often results in high phosphate losses not apparent by the usual “heads minus tails” calculation.

Considerable work was done by the Bureau of Mines and by the writer in 1932, first independently and later jointly,⁵ on the tabling of this mixture of phosphate pebble and quartz in the minus 20-mesh sizes, on the theory that concentrating tables, long on the market, were more appropriate machines than flotation cells for such relatively coarse feed, would eliminate the slime losses due to grinding, and give a lower phosphate loss in the quartz tailings. At the San Gully washer No. 1 of the Southern Phosphate Corporation, a single standard size of concentrating table was installed in 1932 and the practice developed in this pilot operation has determined the design of this company’s Standard washer at Medulla (early 1934), and also its new San Gully No. 2 washer (early

1937) near Lakeland. The latter is the plant that will be described in detail in this paper.

SCREENING AND SCRUBBING THE PLUS 5-MESH PHOSPHATE

The sludge from the mine pit enters a receiving box (1, Fig. 4) about 45 ft. above the ground, where it is divided into two streams (Fig. 5) before flowing through the double-unit washer. Each stream passes over an inclined stationary screen (2, Fig. 4) that is 41 in. wide by 12 ft. long, with slotted openings $\frac{3}{8}$ by $\frac{1}{2}$ in. and a slope of $1\frac{1}{2}$ in. per foot. The oversize flows into two parallel trommels (4) 4 ft. in diameter by 12 ft. long with $1\frac{1}{2}$ in. openings and turning at 19 r.p.m., to remove the large clay balls. About 250 gal. per min. of water under 200-lb. pressure is sprayed against each trommel to prevent clay balls from lodging in the openings. The undersize from the trommels, after further dewatering on stationary screens (5) just underneath the trommels and exactly like the screens at the head of the washer, passes over two 5 by $8\frac{1}{2}$ -ft. Robins vibrating screens (6) equipped with $\frac{3}{8}$ by $\frac{1}{2}$ -in. Tyler Ton-Cap screen cloth. These stationary and vibrating screens eliminate 70 to 80 per cent of the minus 20-mesh material in the matrix.

The oversize from the vibrating screens flows to a double log washer (7). The logs are 8-in. double, extra heavy pipes, 21 ft. long and sloped $1\frac{3}{16}$ in. per foot. The speed is 27 r.p.m. About 200 gal. per min. of water is added to each pair of logs. From 10 to 15 min. is taken for the logs to work the material forward to the discharge end, and during that time much of the clay is broken up by the log teeth that plough through the pulp. No overflow is made and the whole product discharges to two 5 by 12-ft. Robins vibrating screens (8) equipped with Tyler Ton-Cap screen cloth with $\frac{5}{32}$ by $\frac{1}{2}$ -in. openings. The oversize from these screens flows to an 8-ft. by 36-in. Hardinge mill (9) which turns at 26 r.p.m. No grinding media are used but flights 6 in. high lift and tumble the material in the mill. Water is added at the rate of 400 to 500 gal. per min. When the feed reaches the Hardinge mill it is fairly clean of clay and the washing is principally rock on rock. The clay that does remain in the feed is quickly worked into suspension by attrition.

The Hardinge mill discharge flows over one 5 by 12-ft. Robins vibrating screen (10) equipped with Tyler Ton-Cap screen cloth with $\frac{5}{32}$ by $\frac{1}{2}$ -in. openings (Fig. 6). The oversize is a clean product (plus 5-mesh phosphate) which discharges into a 20-in. belt elevator (11) and from there onto a 20-in. belt conveyor (12), thence to a 200-ton storage bin (13) for transportation to the drying plant at Ridgewood. The elevator is 40 ft. center to center, vertical and travels at 250 ft. per min. The conveyor is 40 ft. center to center, inclined at 14° to the horizontal and travels at 250 ft. per min. also. The bin is steel, 20 ft. in diameter, with conical bottom.

The five Robins vibrating screens mentioned above are run in a counter-flow direction at 1000 r.p.m. and set at approximately 22° from the horizontal. From 150 to 200 gal. per min. of spray water is discharged onto each screen through Concenco spray nozzles.

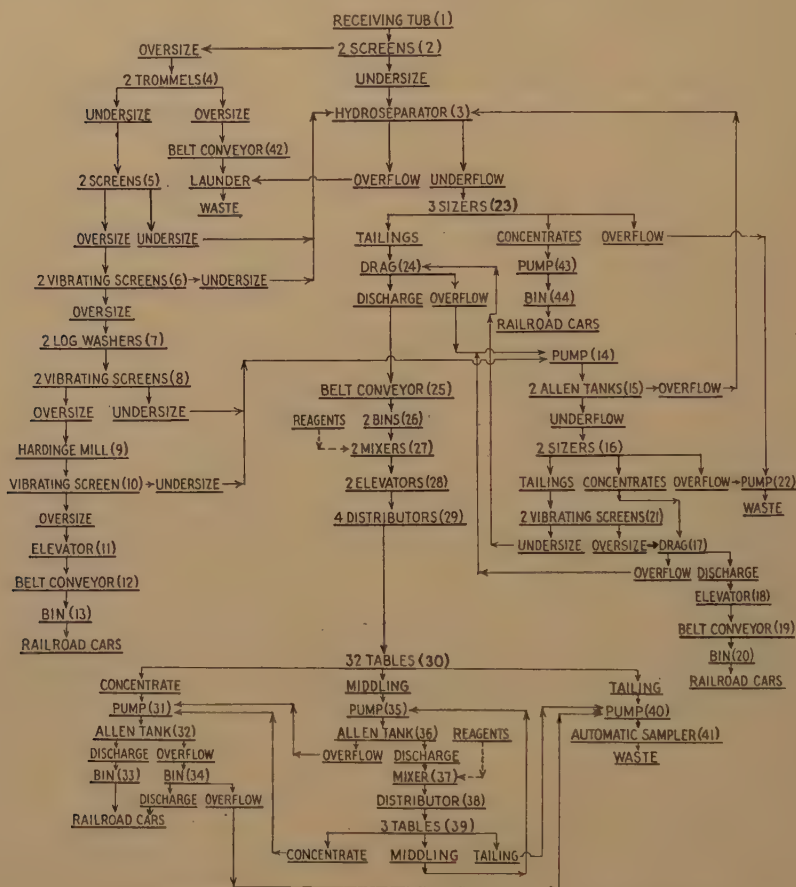


FIG. 4.—FLOWSHEET OF THE SAN GULLY WASHER AND RECOVERY PLANT.
Key on opposite page.

Of the 375 tons of matrix per hour delivered to the washer, approximately 10 tons is recovered as plus 5-mesh phosphate, which analyzes 72 to 75 per cent bone phosphate of lime. The undersize of the $\frac{3}{4}$ -in. dewatering screens is about 300 tons. The clay balls rejected to waste are about 5 tons and the undersize of the $\frac{5}{32}$ -in. screens is about 60 tons.

CLASSIFYING THE MINUS 5 PLUS 14-MESH PHOSPHATE

The undersize of the three vibrating screens that are equipped with $\frac{5}{32}$ -in. screen cloth flows to a sump underneath the Hardinge mill and is

pumped to a receiving box about 35 ft. above the ground, by an 8-in. Georgia Iron Works sand pump (14) with 28-in. diameter runner, directly driven by a 100-hp., 600-r.p.m., General Electric slip-ring motor. The undersize is split as it flows from the receiving box, to two 10-ft. square Allen sand tanks (15), where fine sands, slime and water overflow; and a spigot product at 65 to 70 per cent solids discharges to two six-pocket steel Fahrenwald sizers (hydraulic classifiers) (16).

The hydraulic water to each sizer is 1200 gal. per min., furnished by a low-pressure pump. A constant-level tank is not essential when low-pressure water is used. By proper adjustment of the hydraulic water and the discharge mechanism, well graded and slime-free products are

FIG. 4.—KEY TO FLOWSHEET OF SAN GULLY WASHER AND RECOVERY PLANT.

1. Receiving tub, 4 by 6 by 12 ft.
2. Stationary screen, 41 in. by 12 ft.; slope $1\frac{1}{2}$ in. per ft.; slotted openings, width $\frac{3}{64}$ in.
3. Dorr hydroseparator, 40-ft. diameter.
4. Circular trommel, 4 by 12 ft.; 19 r.p.m.; $1\frac{1}{2}$ -in. round openings.
5. Stationary screen, 41 in. by 14 ft.; slope $1\frac{3}{4}$ in. per ft.; slotted openings, width $\frac{3}{64}$ in.
6. Robins Gyrex screen, 60 by 102 in.; screen openings, $\frac{3}{64}$ in.
7. 21-ft. double log washer, pipe type; slope $1\frac{3}{16}$ in. per ft.; 27 r.p.m.
8. Robins Gyrex screen, 60 by 144 in.; 5-mesh screen.
9. Hardinge mill, 8 ft. by 36 in., with flights 6 in. high; 26 r.p.m.
10. Same as 8.
11. Bucket elevator; 20-in. belt; buckets, $6\frac{1}{2}$ by $6\frac{3}{4}$ by 16 in.; 250 ft. per min.
12. Belt conveyor; width, 20 in.; 250 ft. per min.
13. Bin, 20-ft. diameter; 200 tons capacity.
14. 8-in. Georgia Iron Works centrifugal pump; 100-hp. motor.
15. 10-ft. Allen tank.
16. Six-pocket Fahrenwald sizer.
17. Dewatering drag, Esperanza type; 5 by 25 ft.
18. Bucket elevator; 20-in. belt; buckets, $6\frac{1}{2}$ by $6\frac{3}{4}$ by 16 in.; 300 ft. per min.
19. Belt conveyor; width, 24 in.; 250 ft. per min.
20. Same as 13.
21. Link-Belt vibrating screen, 4 by 5 ft.; 16-mesh screen.
22. 12-in. Georgia Iron Works centrifugal pump; 200-hp. motor.
23. Same as 16.
24. Same as 17.
25. Belt conveyor; width, 24 in.; 300 ft. per min.
26. Bin, 22-ft. diameter; 300 tons capacity.
27. Denver conditioner, 5 by 5 ft.
28. Bucket elevator; 20-in. belt; buckets, $6\frac{1}{2}$ by $6\frac{3}{4}$ by 16 in.; 250 ft. per min.
29. Eight-way Concenco revolving feed distributor.
30. No. 6 Deister-Overstrom diagonal-deck table.
31. 8-in. Wilfley centrifugal pump; 100-hp. motor.
32. Same as 15.
33. Same as 13.
34. Same as 13.
35. 4-in. Wilfley centrifugal pump; 30-hp. motor.
36. Same as 15.
37. Denver conditioner, 3 by 3 ft.
38. Three-way Concenco revolving feed distributor.
39. Same as 30.
40. 8-in. Georgia Iron Works centrifugal pump; 75-hp. motor.
41. General Engineering Co. electric sampler.
42. Belt conveyor; width, 20 in.; 250 ft. per min.
43. Same as 35.
44. Same as 13.

discharged at 60 per cent solids from the six pockets. The overflow, containing slime and minus 65-mesh sands, passes to waste. The first three spigots discharge 80 per cent of the minus 5 plus 14-mesh phosphate that was in the feed as a finished product to a 5 by 25-ft. belt drag classifier (17) of local design for dewatering (Fig. 7). The discharge from the last three spigots contains approximately 20 per cent minus 14-mesh and is passed over two 4 by 5-ft. Link-Belt vibrating screens with openings equivalent

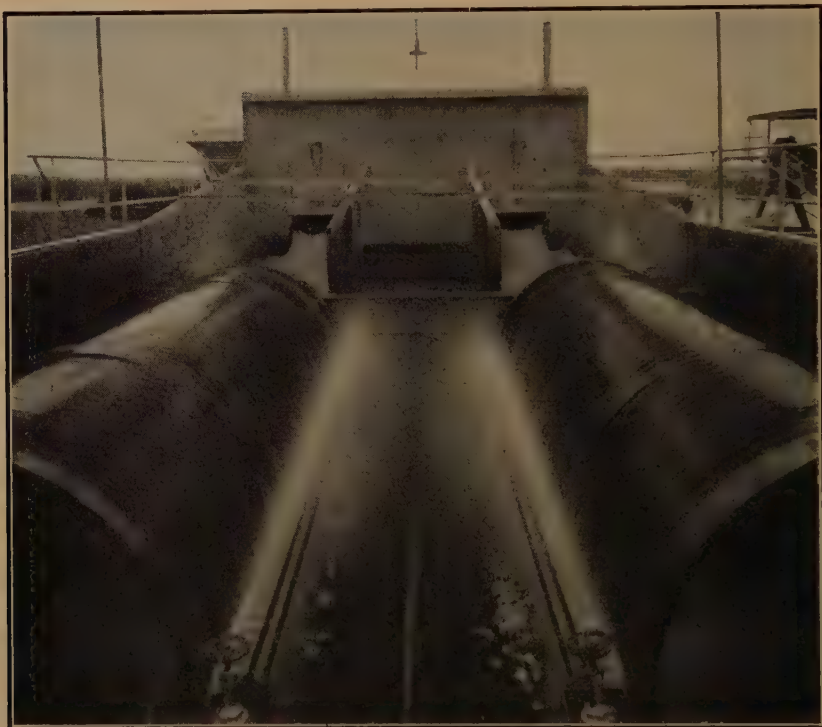


FIG. 5.—RECEIVING END OF SAN GULLY WASHER.

to 16-mesh Tyler standard. The oversize is a finished product and flows into the drag classifier along with the products of the first three spigots.

The minus 5 plus 14-mesh discharge product of the drag is conveyed by a 20-in. belt elevator (18) and a 24-in. belt conveyor (19) to a steel bin exactly like the plus 5-mesh storage bin to await shipment to Ridgewood. The elevator is 40 ft. center to center, vertical, and travels at 250 ft. per minute. The conveyor travels at 250 ft. per minute, is 70 ft. center to center, and inclined at 14° to the horizontal.

CLASSIFYING THE MINUS 14-MESH MATERIAL

In the classification of the minus 14-mesh material, which averages 300 tons per hour, a split is made at 20 mesh and at 65 mesh. The plus

20-mesh fraction, which is 8 to 10 tons, analyzes from 70 to 75 per cent bone phosphate of lime and is removed as a finished product. The minus 20 plus 65-mesh sands, which is 60 to 80 tons, analyzes about 30 per cent bone phosphate of lime and is sent to storage bins for feed to the table plant. The minus 65-mesh material is too low grade for treatment and is rejected to waste.

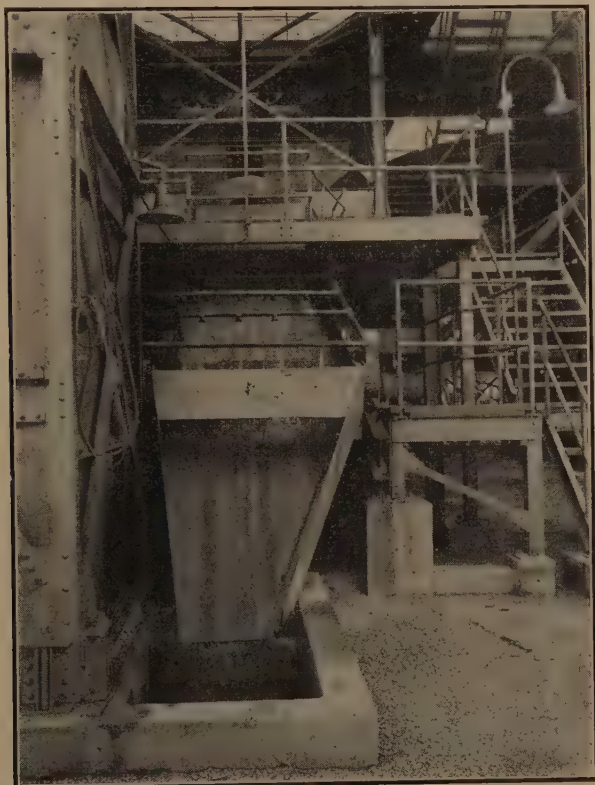


FIG. 6.—DISCHARGE END OF SAN GULLY WASHER.

The undersize from the screens with $\frac{3}{64}$ -in. openings and the overflow from the two Allen tanks, containing 300 tons of minus 14-mesh material per hour in 6000 gal. per min. of water, flow into a 40-ft. Dorr hydro-separator. The speed of the rakes is 1 r.p.m. The overflow to waste is 5000 gal. per min., which contains 150 tons or more of minus 65-mesh material. The spigot discharge of 1000 gal. per min. containing 150 tons of solids per hour is fed to three six-pocket wood Fahrenwald sizers.

The feed rate to each of the three wood sizers is 50 tons per hour of solids in a pulp of 300 to 400 gal. per min. The hydraulic water to each sizer is 800 gal. per min., furnished by the same low-pressure pump that furnishes water to the two steel sizers. The overflow, containing the

minus 65-mesh sands flows to waste. The first two spigots discharge the plus 20-mesh phosphate that was in the feed as a finished product, which is pumped to a 200-ton storage bin by a 4-in. Wilfley sand pump with 16-in. diameter runner and driven at 1200 r.p.m. The last four spigots discharge the minus 20 plus 65-mesh phosphatic sands in the feed to a drag classifier exactly like the one that dewateres the minus 5 plus 14-mesh phosphate. The overflow from both dewatering drags goes by gravity to the sump underneath the Hardinge mill for dilution water. The undersize from the two Link-Belt screens with 16-mesh screen cloth flows into the drag along with the last four spigots of the sizers. The dewatered product of minus 20 plus 65-mesh sands falls onto a 24-in. belt

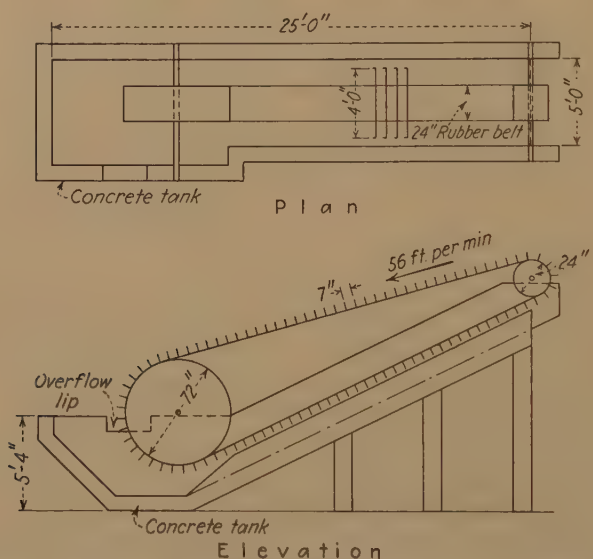


FIG. 7.—SKETCH OF BELT DRAG CLASSIFIER FOR DEWATERING.

conveyor, which delivers it to two 300-ton storage bins for feed to the table plant. The conveyor travels at 300 ft. per minute, is 165 ft. center to center and is inclined at 18° to the horizontal. The two storage bins are similar to the others mentioned except that they are 22 ft. in diameter instead of 20 feet.

CONCENTRATING THE MINUS 20 PLUS 65-MESH PHOSPHATIC SANDS

The phosphatic sands are concentrated by oiling the phosphate particles to make air bubbles stick to them, then passing the oiled feed over regular ore-dressing tables. The oiled phosphate particles with air bubbles attached float over the side of the tables and away from the quartz. A middling is made, which consists mostly of unoled phosphate particles and quartz. The middling is oiled in a separate conditioner,

after which the phosphate is recovered on tables similar to those that treat the original feed.

The table feed is discharged from the two 300-ton bins through 4-in. iron stopcocks to two 5 by 5-ft. Denver conditioners (27). Reagents are fed by gravity from three 300-gal. tanks to the conditioners at the rate of $1\frac{1}{2}$ to 2 lb. of fatty acid, 2 to 4 lb. of light-gravity fuel oil and $\frac{1}{2}$ lb. of caustic soda per ton of feed. Water is added to dilute the feed pulp to 50 to 60 per cent solids. The mixing is done by manganese-steel propellor that turns at the rate of 400 revolutions per minute.

The fatty acid and light-gravity oil are received in tank-car shipments and unloaded by a 3-in. Tuthill oil pump into 15,000-gal. storage tanks at ground level. The reagents are delivered to the 300-gal. tanks by two $\frac{3}{8}$ -in. Tuthill oil pumps. The caustic soda is received in the solid form (700-lb. drums) in carload shipments. Drums are mixed one at a time as needed to form a 10 per cent aqueous solution by a "Lightnin" portable mixer and delivered to a 300-gal. tank from the mixing tank by gravity.

The conditioned pulp is elevated by two 20-in. belt elevators to four 4-in. pipe launders with a slope of 3 in. per foot, and then conveyed into the table building to four eight-way Concenco revolving feed distributors (29). The elevators are 36 ft. center to center, inclined at 25° to the vertical and travel at 250 ft. per min. Because of packing in the bottom of the buckets, spray water under 80-lb. pressure is used at the point of discharge for satisfactory dumping.

The table building is 52 ft. wide and 170 ft. long, by 12 ft. high at the eaves, with structural-steel frame resting on concrete foundations, and No. 24 gauge galvanized copper-bearing steel. The floor is concrete. On one end is a control room of all the electrical equipment. Remote-control switches for all the motors are arranged on a switchboard, which is in plain view from the inside of the table building. There is a switch at each motor as well.

Pipe launders (3 in. in diameter) at a slope of 3 in. per foot convey the feed from the distributors to 32 No. 6 Deister-Overstrom diagonal-deck tables (30), supported on concrete foundations 3 ft. above the concrete floor (Fig. 8). Water is added to the feed boxes to dilute the feed to 30 per cent solids. Deck covering is linoleum. Oak riffles, $\frac{5}{16}$ in. wide and tapering from $\frac{1}{4}$ in. at the head-motion end to $\frac{1}{8}$ in. at the terminal end, are spaced $1\frac{1}{2}$ in. apart. Every eighth riffle is a pool riffle and $\frac{1}{16}$ in. higher than the others. The riffles in the dressing zone are $\frac{1}{16}$ in. high, untapered. Each table is run at 285 r.p.m. with a $\frac{3}{4}$ -in. stroke. The transverse slope is about $\frac{3}{4}$ in. per foot.

Small particles of the oil-conditioned feed stick to the pipe launders and to the linoleum covering of the tables. To keep the particles from building up and causing trouble, the table decks are scrubbed once every 4 hr. with a coarse fiber brush, while the tables are in motion, and the

pipes are cleaned once a week by a Lagonda water-driven tube cleaner. Experiments have been made with several deck coverings, but none has been found to which none of the table feed would stick. A Monel metal covering seems to be slightly better than the linoleum, in that the sticking particles brush off more easily.

When the feed is tabled, the phosphate particles that were selectively oiled and agglomerated in the conditioners stratify on the tables on top of the quartz. After proper adjustment of the transverse slope and of the ore-dressing water, the phosphate washes across the riffles and



FIG. 8.—INSIDE VIEW OF TABLE BUILDING.

discharges rapidly along the tailing side as a finished product. The quartz, which is unoiled, is carried to the concentrate end by the riffles and discharged as waste tailing. A middling cut of 15 to 20 per cent of the feed is made along 6 to 10 in. of the concentrate end that is nearest the tailing side. Open launders (8 by 10 in.) at a slope of 2 in. per foot carry these products to separate concrete sumps located in the center of the building.

The finished concentrate is pumped at 10 per cent solids to the outside of the table building to a 10-ft. Allen sand tank for dewatering by an 8-in. Wilfley sand pump with 18-in. diameter runner, and driven at 1200 r.p.m. The dewatered concentrate discharges at 65 to 70 per cent solids into a 200-ton storage bin just below the Allen tank, to await transportation to Ridgewood. The concentrate is hauled to the drying plant

in Atlantic Coast Line bottom-dump railroad cars, the floor sections of which fit together loosely. To prevent the loss of concentrate, the bottoms of the cars are lined with burlap sacks, which, incidentally, act as an excellent filter for further dewatering. Water drains rapidly from the cars during loading.

The middling, of quartz and unagglomerated phosphate grains, is pumped at 25 per cent solids from the middling sump to the outside of the table building to a 10-ft. Allen tank for dewatering by a 4-in. Wilfley sand pump with 16-in. diameter runner, and driven at 1200 r.p.m. The dewatered middling is conditioned with reagents in a 3 by 3-ft. Denver conditioner and then conveyed by gravity through a 4-in. pipe launder to a three-way Concenco revolving feed distributor in the table building. From the distributor the middling is conveyed by pipe launders (3-in. diameter) at a slope of 3 in. per foot, to three No. 6 Deister-Overstrom diagonal deck tables. Finished concentrate and waste tailing join the concentrate and tailing of the other tables. A small middling from these three tables returns to the middling circuit. No built-up condition is obtained because after the phosphate becomes oiled it is recovered as a concentrate.

Despite baffling in the Allen tanks, phosphate is floated over the overflow from the middling and the finished concentrate. These overflows run by gravity into a 200-ton storage bin, which is baffled to settle out all the phosphate. The bin overflow goes by gravity to the tailing sump for dilution water. The phosphate is loaded into cars from time to time as it accumulates.

The waste tailing is pumped at 10 per cent solids to the outside of the table building, to a wooden launder that takes away the hydroseparator overflow, by an 8-in. Georgia Iron Works pump with 24-in. diameter runner, and driven at 600 r.p.m. Primary and secondary control equipment provides variable motor speed to 50 per cent reduction.

TAILINGS DISPOSAL AND WATER SUPPLY

The plus 1½-in. clay balls that are rejected by the two trommels at the head of the washer drop onto a 20-in. belt conveyor, which delivers them to the wooden launder that removes the hydroseparator overflow. The conveyor is 40 ft. center to center, level, and travels at 250 ft. per minute. The hydroseparator overflow, together with the clayballs and the tailing of the table plant, flow by gravity in a wooden launder with ⅛ in. slope per foot to a mined-out area near the washer. The overflow of the five Fahrenwald sizers is pumped to this mined-out area by a 12-in. Georgia Iron Works pump with 36-in. diameter runner driven at 600 r.p.m. Reduction to 50 per cent of the motor speed is provided by primary and secondary control equipment.

The tailings water is circulated over a wide area to allow the sands and slime to settle out, and then returned by dams and ditches to the pumping station at the washer to be used over again. Normally, rainfall is sufficient to maintain the water supply but during a long dry period a deep-well pump is operated to deliver water into the circulating system.

The pumping station contains five Allis-Chalmers pumps. A two-stage centrifugal pump, 5000 gal. per min. at 200-lb. pressure, direct-driven by a 750-hp., 2200-volt, 1200-r.p.m., synchronous motor, furnishes the water to the mine pit and to the trommels at the head of the washer. A single-stage pump, 2000 gal. per min. at 180-ft. head, direct-driven by a 150-hp., 2200-volt, 1800-r.p.m. induction motor, furnishes the water to the washer exclusive of the trommels mentioned above and the Fahrenwald sizers. Water to the sizers and to the table plant is furnished by two pumps with a total capacity of 8000 gal. per min. One of these is a single-stage pump, 5000 gal. per min. at 50-ft. head, direct-driven by a 75-hp., 2200-volt, 900-r.p.m. induction motor. The other is a single-stage pump, 3000 gal. per min. at 50-ft. head, direct-driven by a 50-hp., 2200-volt, 900-r.p.m. induction motor. The fifth pump is a spare to furnish additional ore-dressing water if ever needed. It is a single-stage pump, 1000 gal. per min. at 174-ft. head, direct-driven by a 75-hp., 2200-volt, 1800-r.p.m. induction motor.

PRODUCTION

The washer and table plant produce from 70 to 80 long tons of finished product per hour; 8 to 10 tons of plus 5-mesh, 35 tons of minus 5 plus

TABLE 2.—*Assays and Tons per Hour of the Phosphate Produced Over a Period of a Month*

Product	Long Tons per Hour	Bone Phosphate of Lime, Per Cent	Insoluble, Per Cent
Plus 5-mesh.....	8.89	73.17	11.06
5/4-mesh.....	35.28	75.12	8.96
1 1/2 0-mesh.....	7.48	70.51	15.06
Table-plant concentrate.....	24.19	75.36	8.92

14-mesh, 8 to 10 tons of minus 14 plus 20-mesh, and 20 to 25 tons of concentrate product. This represents a recovery of practically 100 per cent of the plus 20-mesh phosphate and about 75 per cent of the plus 65-mesh phosphate that was in the matrix. Assays and the tons per hour of these products over a period of a month are given in Table 2. Table-plant feed and tailing analyze about 30 per cent and 7 per cent bone phosphate of lime, respectively.

SAMPLING

Grab samples of the three finished products and of the table-plant concentrate are taken with a scoop as the phosphate is loaded into railroad cars. The table-plant feed is sampled by a hand-operated sample cutter at 30-min. intervals as it is discharged from the storage bins to the conditioners. The table-plant tailing is sampled by a General Engineering Co. automatic electric sampler. These samples are taken for each 8-hr. shift and sent to the chemical laboratory of the Southern Phosphate Corporation at Ridgewood for analysis.

MOTORS AND DRIVES

General Electric motors are used exclusively at the washer and table plant. Allis-Chalmers motors are used at the pumping station. The

TABLE 3.—*Data on Motors and Drives*

Duty	Number Used	Motor Hp.	Kind of Drive
Trommels.....	2	5	Gear
Clay-ball conveyor.....	1	5	Gear
5 by 8½-ft. Robins vibrating screens.....	2	5	Gear
Double log washer.....	2	25	Gear
5 by 12-ft. Robins vibrating screens.....	3	7½	V-belt
Hardinge mill.....	1	75	V-belt
Plus 5-mesh elevator.....	1	7½	Gear
Plus 5-mesh conveyor.....	1	5	Gear
8-in. pump to handle undersize of 5 by 12-ft. Robins vibrating screens.....	1	100	Direct
4 by 5-ft. Link-Belt vibrating screens.....	2	3	V-belt
5/14-mesh dewatering drag classifier.....	1	10	Gear
5/14-mesh elevator.....	1	10	Gear
5/14-mesh conveyor.....	1	10	Gear
Dorr hydroseparator.....	1	10	Gear
12-in. pump to handle oversize of Fahrenwald sizers...	1	200	Direct
4-in. pump to handle 1½-mesh.....	1	30	Direct
Table-feed dewatering classifier.....	1	15	Gear
Table-feed conveyor.....	1	15	Gear
¾-in. pumps to feed reagents.....	2	½	Direct
5 by 5-ft. Denver conditioners.....	2	7½	V-belt
3 by 3-ft. Denver conditioner.....	1	3	V-belt
Table-feed elevators.....	2	10	Gear
Table-feed distributors.....	5	½	Gear
Tables.....	35	2	V-belt
8-in. pump to handle concentrate.....	1	100	Direct
4-in. pump to handle middling.....	1	30	Direct
8-in. pump to handle tailing.....	1	75	Direct
General Engineering Co. sampler.....	1	¼	Gear
Pumps for water supply to washer and plant.....	3	150-75-50	Direct

horsepower and kind of drive of each motor is shown in Table 3. The total rated horsepower is 1182.

The gear motors for the dewatering drag classifiers, conveyors and elevators are special purpose and splashproof. The gear motors for the trommels and the hydroseparator are special purpose and totally enclosed. The motors for the Robins vibrating screens are totally enclosed. All other motors are open.

BELT CONVEYORS AND ELEVATORS

The conveyors are made up of 24-in. diameter pulleys that are split and double-armed, five-ply Goodyear conveyor belt, Link-Belt three-roll antifriction troughing idlers and self-aligning troughing idlers, and Link-Belt single-roll antifriction return idlers.

The elevators are made up of 30-in. diameter boot pulleys, 36-in. diameter head pulleys, six-ply Goodyear elevator belt, and $6\frac{1}{2}$ by $6\frac{3}{4}$ by 16-in. heavy seamless buckets spaced 12 in. apart. The pulleys are split and double-armed.

TRANSPORTATION AND USES

The wet-rock production of the San Gully washer No. 2 is moved daily by the Atlantic Coast Line R.R. a distance of about 10 miles, and that of the company's Standard plant about 6 miles, to its drying plant at Ridgewood, about 5 miles east of Bartow, the county seat of Polk County, Florida.

This is the permanent plant from which the finished phosphate rock is shipped to the consuming centers of the United States and to the principal markets in Canada, Europe and Japan, via Port Tampa, where phosphate boats are loaded by the Atlantic Coast Line R.R. at the rate of 1200 tons per hour, by means of specially designed elevators.

About 90 per cent of the total phosphate rock produced is used in making "superphosphate," one of the principal ingredients in commercial fertilizers.

ACKNOWLEDGMENTS

The writer wishes to express his appreciation to Mr. Chester A. Fulton and Mr. E. A. Pierce, President and Vice-President of Southern Phosphate Corporation, respectively, for their interest and cooperation.

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DISCUSSION

(Paul M. Tyler presiding)

F. A. JORDAN, Youngstown, Ohio.—I would like to ask a little about the tabling of the minus 20 plus 65-mesh material. Do you limit it to that size because at a larger size the material is made up of aggregates of the two, or is it because that is the largest size at which the scheme can be worked?

J. W. PAMPLIN.—That is the size at which the quartz sand begins to dilute the phosphate pebble at minus 20 mesh. The plus 20 mesh we recover by screening, as there is no coarse sand in that. The minus 65-mesh material analyzes about 10 per cent bone phosphate of lime, and we consider it tailing.

F. A. JORDAN.—Let me put the question in another way. I have in mind another problem. Suppose that you could make the separation at larger than 20 mesh? At 14 mesh or 10 mesh? Would that scheme work?

J. W. PAMPLIN.—Yes, I think so. In the experimental work we did some tabling of the size of products coarser than 20 mesh, but, of course, the product itself analyzed from 65 to 70 per cent B.P.L. because there was not much quartz in it.

J. C. WILLIAMS,* Westport, Conn.—In Table 1, the slime and the minus 100 per cent together are about 39 per cent of the total matrix with a total of over 25 per cent of the total B.P.L. How about the recovery of that amount of phosphate?

J. W. PAMPLIN.—That screen analysis is typical of our matrix, and all of that phosphate is lost in the slime. That is a considerable loss as showing 22.8 per cent by weight of slime analyzing 27.83 per cent dry calcium phosphate in the slime analysis and the percentage will vary from time to time, but from 15 to 20 per cent is a good average. It analyzes as high as 35 or 40 per cent B.P.L. at times and then drops down to 15.

O. C. RALSTON,† New Brunswick, N. J.—The plant described is exceedingly well groomed, well designed, and worth seeing. I particularly want to call attention to Fig. 8. The floor is not crowded by the two rows of tables. It is made up of an iron grill. All of the services in the pit below are well lighted and visible from the operator's floor. It is impossible for some fluid to spill or something to stop without being seen. There are many conveniences of that kind. Incidentally, this table plant, I think, represents the largest installation of agglomerate tabling in the world. That is a new type of venture to us, to make use of the agglomerated materials so that cross-water on the table can wash them off while unagglomerated material stays in the riffles. This is not mentioned in the paper, but I think it is worthy of note.

* Flotation Engineer, The Dorr Company.

† U. S. Bureau of Mines.

C. E. LOCKE,* Cambridge, Mass.—In Table 1, some free silica is shown as coarser than 20 mesh. Does that become contaminated in the final phosphate product?

J. W. PAMPLIN.—That is included in the phosphate product. That is not removed.

C. E. LOCKE.—Why was the tabling process adopted instead of flotation?

J. W. PAMPLIN.—In our San Gully matrix the table material is relatively coarse. If we should have minus 65 plus 100 and minus 100 mesh containing a goodly percentage of phosphate material, there would be more consideration given to flotation.

H. M. LAWRENCE,† New York, N. Y.—In 1927, when I was first assigned by the Bureau of Mines to the Florida phosphate problem, I made a number of trips to Florida. Table 2 presents very emphatically the advances since that time. There could be a number of different viewpoints as to whether or not all of the 14 to 20-mesh material, there shown as $7\frac{1}{2}$ tons an hour, was lost in the old washers, but there is no doubt that there was none of what is now called table-plant concentrate recovered. All of that phosphatic sand was lost. Discussions with plant managements in Florida at first brought the almost inevitable "Why should we try to save this material? We have hundreds of square miles of phosphates. Why bother trying to save the fine sizes?" The discussions in the early days were predicated on more careful attention to washing details and investigations that might recover some of the other material that was not susceptible to recovery by the customary washing procedures. Now 24 tons is recovered each hour as table-plant concentrate. Naturally, that amount would vary, depending upon the particular deposit as well as upon variations in the individual deposits. Nearly 50 per cent additional recovery has been made through advances in technology, many of which were developed first in other fields but not applied in Florida until 1927 or thereafter.

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Phosphate Fertilizers by Calcination Process— Volatilization of Fluorine from Phosphate Rock at High Temperatures

BY K. D. JACOB,* D. S. REYNOLDS* AND H. L. MARSHALL*

(New York Meeting, February, 1936)

ALL types of commercial phosphate rock produced throughout the world contain fluorine in quantities ranging from approximately 0.4 to 1.3 per cent in the Curaçao and Christmas Island phosphates to 3.1 to 4.2 per cent in the phosphates of the United States and North Africa¹. For a given type of rock, the fluorine content is usually roughly proportional to the phosphorus content.

SIGNIFICANCE OF FLUORINE IN PHOSPHATE ROCK

On the basis of chemical and X-ray diffraction studies, Hendricks and co-workers² concluded that fluorapatite, $\text{Ca}_{10}\text{F}_2(\text{PO}_4)_6$, is the essential phosphatic constituent of the commercial types of domestic phosphate rock. On the other hand, Bredig and co-workers³ have recently advanced the opinion, also based on X-ray diffraction studies, that the mineral phosphates usually contain carbon dioxide in the apatite lattice, and that the phosphate rocks of the United States are really fluorcarbonate apatites. However, the domestic types of phosphate rock do not seem to differ greatly from coarsely crystalline fluorapatite in their behavior toward chemical reagents in general, and for the purpose of this paper it will suffice to consider fluorapatite as being the essential phosphatic constituent of these types of rock.

The available data¹ indicate that the different commercial types of domestic phosphate rock usually contain about 12 to 52 per cent more fluorine than theoretically is required to form fluorapatite with all the phosphorus. Calcium fluoride (fluorite) has been definitely identified in only a very few of the samples that have been examined^{1,2}. Bredig and co-workers⁴ state that fluorapatite can take into its crystal lattice excess calcium fluoride to the extent of approximately 70 per cent of that theoretically required by the formula $\text{Ca}_{10}\text{F}_2(\text{PO}_4)_6$ without causing more

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¹ References are at the end of the paper.

than a very slight broadening of the lattice. The same authors offer this as an explanation of the presence of the excess fluorine occurring in most types of phosphate rock. However, the presence of fluorine in excess of that required by the fluorapatite equivalent of the total phosphorus does not seem to interfere seriously in the preparation of fertilizers from phosphate rock.

The comparatively low citrate-solubility¹ and fertilizer efficiency^{5,6} of many types of raw phosphate rock seem to be attributable, at least in part, to the presence of the phosphorus in the form of fluorapatite. Because of the deleterious effect of fluorine on the health of animals⁷, its presence in phosphate rock has prevented the general use of this cheap source of calcium and phosphorus in mineral mixtures for livestock feeding; removal of the fluorine from phosphate rock is of interest in this connection. Ordinary superphosphate, which has been the principal phosphate fertilizer manufactured in this country since the beginning (about 1850) of the domestic chemical fertilizer industry, contains about 1.3 to 1.8 per cent of fluorine. A considerable portion of the domestic supply of fluosilicates and fluosilicic acid is derived from the fluorine volatilized (about 30 per cent of the fluorine in the phosphate rock) during the manufacture of ordinary superphosphate.

There is evidence that fluorine compounds may be toxic to plants under certain conditions, whereas under other conditions they either may have no effect or may stimulate plant growth to some extent. Bartholomew⁵ has shown that fluorine accumulates principally in the roots of plants and that high concentrations of fluorine occur in the aerial portions only when excessively large quantities of fluorine compounds are applied to the soil. Also, long-continued application of raw phosphate rock and superphosphate has not significantly increased the fluorine content of the aerial portions of plants grown on permanent soil-fertility plots in different sections of the country⁸. However, there is evidence that fertilization of soils with fluorine-bearing phosphates may appreciably increase the fluorine content of drainage water, and Hart and co-workers⁹ have raised the question as to whether this may ultimately increase the fluorine content of drinking water sufficiently to contribute to the incidence of mottled enamel of human teeth.

In view of these facts it seems desirable to investigate methods for the removal of fluorine from phosphate rock in relation to the possible development of an economical process for the manufacture of a low-fluorine phosphate suitable for use both as a fertilizer and as a mineral supplement in livestock feeding. Accordingly, this paper presents data on the factors affecting the volatilization of fluorine and the production of citrate-soluble phosphate by heating phosphate rock in the presence of water vapor.

MATERIALS AND EXPERIMENTAL PROCEDURE

The phosphate rocks were ground to pass a 40-mesh sieve, and the 40 to 80-mesh particles were used. Partial chemical analyses of the samples are given in Table 1. In the last three columns of this table, and elsewhere in this paper, F_A represents the fluorine in excess of that corresponding to the fluorapatite, $\text{Ca}_{10}\text{F}_2(\text{PO}_4)_6$, equivalent to the total phosphorus in the rock. F_B and F'_B represent the fluorine corresponding to the respective atoms of fluorine in the fluorapatite equivalent of the total phosphorus; for convenience these are designated as the first (F_B) and second (F'_B) atoms of fluorine. This distribution of the fluorine is based on the assumption that all the phosphorus is present in the form of either fluorapatite or a fluorine-bearing phosphate in which the ratio of fluorine to phosphorus is the same as that in fluorapatite.

TABLE 1.—*Composition of Phosphates*

Type and Source of Phosphate	P_2O_5 , Per Cent		Total F, Per Cent	SiO_2^a , Per Cent	Total F, Per Cent, as:		
	Total	Citrate-soluble			F_A	$F_A + F_B$	F'_B
Florida land pebble ^b	30.47	4.20	3.67	9.96	26.0	63.0	37.0
Florida hard rock, Dunnellon...	35.25	3.83	3.62	4.69	13.2	56.6	43.4
Tennessee brown rock, Wales...	34.64	1.99	3.67	5.30	15.8	57.9	42.1
Tennessee blue rock, Gordonsburg.....	31.69	2.36	3.65	6.48	22.6	61.3	38.7
Tennessee white rock, Godwin...	35.46	3.69	3.66	2.55	13.6	56.8	43.2
South Carolina land rock, Johns Island.....	26.45	5.06	3.41	13.08	30.8	65.4	34.6
Idaho, Conda.....	33.56	3.73	3.37	3.93	11.2	55.6	44.4
Montana, Garrison.....	28.96	1.99	2.96	22.01	12.8	56.4	43.6
Wyoming, Cokeville.....	31.29	1.08	3.54	5.00	21.2	60.6	39.4
Morocco.....	34.63	5.49	3.99	0.74	22.6	61.3	38.7
Nauru Island.....	38.38	5.03	2.68	0.39 ^c	0	36.2	63.8
Fluorapatite, Quebec Province, Canada.....	39.38	0.85	3.03	1.25	0	42.1	57.9
Fluorapatite, Amherst County, Virginia.....	39.46	2.48	3.04	3.96	0	42.1	57.9

^a Total silicon.

^b From a deposit of unknown location.

^c Total material insoluble in 1:1 hydrochloric acid.

The apparatus and general experimental procedure have been described in previous papers^{9,10}. The charge was heated in a small platinum boat in an electric resistance furnace of the horizontal tube type. The cold charge was placed in the furnace at the initial temperature of the experiment. The boat was withdrawn from the furnace

at the maximum temperature of the experiment and was allowed to cool rapidly in the laboratory atmosphere. The flow of air through the furnace was maintained at 120 c.c. per minute (dry air measured at 25° C.). In order to insure the presence of an ample quantity of moisture, water was supplied to the furnace atmosphere at the rate of 0.8 gram per minute.

All analytical samples were ground to pass a 100-mesh sieve. Fluorine was determined by the Willard and Winter method^{11,12}. Citrate-insoluble phosphorus was determined by the recently adopted modification¹³ of the official neutral ammonium citrate method; the digestions were made in the presence of filter-paper pulp in the ratio of 1.0 gram of sample to 100 c.c. of citrate solution.

Effect of Furnace Atmosphere.—Reynolds and co-workers^{9,10} have shown that only a small portion of the fluorine is volatilized and little or no increase in the citrate-solubility of the phosphorus is obtained when Florida pebble phosphate is heated in a dry atmosphere at 1400° C. However, when this rock is heated in the presence of an ample supply of water vapor, the other conditions being the same as before, nearly complete volatilization of the fluorine occurs and 90 per cent or more of the phosphorus is converted into the citrate-soluble condition. The results of experiments in small internally fired rotary kilns indicate that sufficient water vapor for the reaction is supplied by the combustion of the fuel (fuel oil). Under otherwise comparable conditions substitution of either nitrogen or oxygen for air in the furnace has no effect on the reaction.

Effect of Particle Size.—In experiments¹⁰ with Florida pebble phosphate at 1400° C., satisfactory and equally good results were obtained with -20, -40, and 40 to 80-mesh materials. In the -20 and -40-mesh materials, 78 and 49 per cent, respectively, of the particles were coarser than 80 mesh. Poor results were obtained with -10, 10 to 20, 20 to 40, -80, 80 to 200 and -200-mesh samples. The products obtained from samples finer than 80 mesh were fused or semifused, whereas those from the coarser materials usually were only caked or sintered. The results obtained with other types of domestic phosphate rock also depended to a certain extent upon the particle size of the material. The completeness of the reaction seems to depend largely upon the extent to which water vapor can diffuse into the interior of the individual particles, the porosity of which is increased by volatilization of the carbon dioxide and by decomposition of the organic matter. As the coarseness of the material is increased beyond a certain limit, it becomes increasingly difficult for water vapor to reach the interior of the particles. On the other hand, as the fineness of the material is increased beyond a certain limit, the particles show an increasing tendency to coalesce at high temperatures into a more or less homogeneous, fused or semifused mass, thereby reducing the surface of contact with the furnace atmosphere and hindering diffusion of water vapor into the charge. The effects of particle size, particularly

with fine material, would probably be less pronounced in a rotary-kiln operation where the charge is subjected to continuous agitation.

Effect of Temperature.—As shown in Table 2, upwards of 95 per cent of the total fluorine was volatilized, and more than 80 per cent of the phosphorus was converted into the citrate-soluble condition when 2.5-gram charges of 40 to 80-mesh rock were heated for 30 min. at 1375° to 1425° C. in the presence of water vapor. Partial volatilization of the fluorine occurred at temperatures below 1300° C. but high percentages of citrate-soluble phosphorus were obtained only at temperatures higher than 1300° C. The poorer results obtained at temperatures above 1425° C. were probably caused by the more rapid softening and coalescing of the rock particles, thereby hindering diffusion of water vapor into the interiors of the comparatively thick charges and reducing the rate of reaction.

TABLE 2.—*Effect of Temperature*

2.5-gram charges of 40 to 80-mesh Florida land pebble heated for 30 minutes in water vapor (0.8 gram per minute) and air (120 c.c. per minute)

Temperature, Deg. C.	Composition of Product, Per Cent		Total F Volatilized, Per Cent	F/B Volatilized, Per Cent	Citrate-solu- bility of P_2O_5 , Per Cent
	Citrate-soluble P_2O_5	F			
1100	1.9	2.26	43.1	0	5.8
1200	3.6	1.59	60.5	0	10.7
1300	8.0	0.63	84.6	58.4	23.6
1350	18.7	0.35	91.4	76.8	55.0
1375	28.7	0.13	96.8	91.4	84.4
1400	31.9	0.02	99.5	98.6	93.5
1425	32.4	0.05	98.8	96.8	95.0
1450	25.1	0.38	90.7	74.9	73.8
1475	25.1	0.33	92.3	79.2	73.8

When the rock was first defluorinated by heating in the presence of water vapor at 1400° C., the citrate-solubility of the phosphorus (93.5 per cent at 1400° C.) was not affected significantly by further heating at temperatures up to 1500° C. The products obtained at 1450° and 1475° C. were semifused and at 1500° C. completely fused.

EXPERIMENTS WITH DIFFERENT PHOSPHATE ROCKS

When 2.5-gram charges (40 to 80-mesh material) of commercial grades of Florida land-pebble, Tennessee brown-rock, Tennessee blue-rock, Idaho, Montana and Wyoming phosphates were heated for 30 min. at 1400° C. in the presence of water vapor, more than 97 per cent of the total fluorine was volatilized and upwards of 90 per cent of the phosphorus was converted into the citrate-soluble condition (Table 3). With the excep-

tion of the Montana phosphate, these materials contained about 4.0 to 11.5 per cent of total silica (Table 1). With the Canadian fluorapatite, Nauru Island, and Morocco phosphates, none of which contained more than 1.25 per cent of silica, less than 70 per cent of the total fluorine was volatilized and not more than 23 per cent of the phosphorus was converted into the citrate-soluble condition; however, good results were obtained with these rocks when the silica content of the charge was increased to 10 per cent¹⁴.

TABLE 3.—*Effect of Water Vapor on Different Types of Phosphate Rock*
2.5-gram charges of 40 to 80-mesh rock heated for 30 minutes at 1400° C. in water vapor (0.8 gram per minute) and air (120 c.c. per minute)

Type or Source of Phosphate	SiO ₂ Content of Original Phosphate, Per Cent	Composition of Product, Per Cent		Total F Volatilized, Per Cent	F ₂ Volatilized, Per Cent	Citrate-solubility of P ₂ O ₅ , Per Cent
		Citrate-soluble P ₂ O ₅	F			
Florida land pebble.....	9.96	31.9	0.02	99.5	98.6	93.5
Florida hard rock.....	4.69	30.2	0.22	94.2	86.6	80.1
Tennessee brown rock....	5.30	36.1	0.01	99.7	99.3	97.8
Tennessee blue rock.....	6.48	31.8	0.06	98.5	96.1	91.6
Tennessee white rock....	2.55	30.3	0.17	95.7	90.0	79.3
South Carolina land rock..	13.08	13.1	0.74	80.8	44.5	43.1
Idaho.....	3.93	37.2	0.01	99.6	99.1	97.9
Montana.....	22.01	28.9	0.02	99.2	98.2	95.7
Wyoming.....	5.00	32.2	0.08	97.9	94.7	90.4
Morocco.....	0.74	1.8	1.41	67.5	16.0	4.8
Nauru Island.....	0.39 ^a	2.1	1.23	57.9	34.0	5.0
Fluorapatite, Canada.....	1.25	9.1	1.51	51.0	15.4	22.8
Fluorapatite, Virginia....	3.96	28.7	0.52	83.2	71.0	71.4

^a Total material insoluble in 1:1 hydrochloric acid.

Although poor results were obtained with rocks containing less than about 4 per cent of silica, the presence of more than 4 per cent of silica did not always insure a high volatilization of fluorine and production of citrate-soluble phosphorus, as shown by the results obtained with the South Carolina rock (13.08 per cent silica). In contrast to the other products, which usually were obtained in the form of friable sinters, nearly complete fusion of the South Carolina rock occurred at 1400° C., with the result that penetration of water vapor into the charge and into the individual particles of rock was greatly hindered. That the high silica content was not alone responsible for the fusion of the South Carolina phosphate is indicated by the absence of fusion at 1400° C. of the Montana rock, which contained 22.01 per cent of silica.

It has been shown¹⁰ that, with stationary charges under comparable experimental conditions, the time required for satisfactory reaction at a

given temperature is markedly affected by the weight of the charge or, in other words, by the thickness of the charge. Accordingly, it seems that the maximum rate of reaction would be obtained by heating the rock under conditions that would permit the maximum degree of contact of water vapor with the individual particles. The ideal situation is represented by a particle falling freely through a uniformly heated space. However, heating the particles in a single-grain layer constitutes the nearest approach to such conditions that can be had with stationary charges.

TABLE 4.—*Effect of Heating Phosphate Rock in Single-grain Layers*
0.2-gram charges of 40 to 80-mesh rock heated at 1400° C. in water vapor (0.8 gram per minute) and air (120 c.c. per minute)

Time of Heating, Min.	Composition of Product, Per Cent		Total F Volatilized, Per Cent	F/B Volatilized, Per Cent	Citrato-solubility of P ₂ O ₅ , Per Cent
	Citrato-soluble P ₂ O ₅	F			
FLORIDA LAND-PEBBLE PHOSPHATE					
5.0	16.42	0.36	91.4	75.8	49.0
10.0	20.19	0.14	96.7	90.7	60.1
15.0	23.04	0.06	98.6	96.1	68.4
20.0	24.21	0.07	98.3	95.2	71.8
30.0	27.85	0.04	99.1	97.5	82.0
TENNESSEE BROWN-ROCK PHOSPHATE					
2.5	23.48	0.44	88.7	73.2	64.0
5.0	31.95	0.10	97.4	93.8	86.9
7.5	33.24	0.04	99.0	97.6	90.0
10.0	34.80	0.02	99.5	98.8	94.3

In order to test this hypothesis, experiments were carried out with single-grain layers of 40 to 80-mesh Florida land-pebble and Tennessee brown-rock phosphates. For these experiments it was necessary to prepare a fresh supply of the 40 to 80-mesh Florida pebble; the analysis, which was only slightly different from that (Table 1) of the 40 to 80-mesh material used in the other experiments, was as follows: total P_2O_5 , 30.10; F, 3.77; total SiO_2 , 9.59 per cent. The heating period was measured from the time that the boat was put in place in the furnace, whereas with all other experiments the period was measured from about the third minute after introduction of the boat. With the boats used in these experiments, 0.2 gram of the 40 to 80-mesh rock gave a layer one grain thick when the particles were uniformly distributed over the bottom of the boat. Most of the particles made contact with the surrounding particles, but care was taken to limit the depth of the layer to the thickness (about 0.38 to 0.17 mm.) of a single particle. The results (Table 4) represent

the average of nine runs under each experimental condition, this number of runs being necessary to obtain sufficient material for analysis.

Comparison of the results given in Tables 3 and 4 show that in 30 min. at 1400° C. the volatilization of fluorine from a single-grain layer of Florida pebble was about the same as from a 2.5-gram charge (approximately 0.4 cm. thick), whereas the citrate-solubility of the phosphorus was significantly lower (11.5 per cent) in the thin-layer product. On the other hand, a single-grain layer of Tennessee brown rock gave nearly the same fluorine volatilization and phosphorus solubility in 10 min. as did a 2.5-gram charge in 30 min. With single-grain layers, heating for 5 min. increased the solubility of the Tennessee brown rock to 87 per cent and that of the Florida pebble to only 49 per cent.

RELATION BETWEEN VOLATILIZATION OF FLUORINE AND CITRATE-SOLUBILITY OF PHOSPHORUS

When phosphate rock containing about 4 per cent or more of total silica is heated at 1400° C. under conditions that permit close contact between the silica and phosphate particles—for example, in comparatively thick layers—the citrate-solubility of the phosphorus in the product usually corresponds closely to the percentage of the second (F'_B) atom of fluorine volatilized (Tables 2 and 3, Fig. 1)^{9,10,14}. However, this relationship does not hold, in general, when the rock contains less than about 2 per cent of silica (Table 3) or when a silica-rich rock is heated at 1350° C. and lower temperatures (Table 2)^{10,14}.

There is evidence that hydroxyapatite [$Ca_{10}(OH)_2(PO_4)_6$], a compound that has a low citrate-solubility, is formed as an intermediate product in the high-temperature reaction between water vapor and phosphate rock. In the presence of water vapor silica-free hydroxyapatite is stable at 1400° C., but in intimate contact with silica it is converted into the citrate-soluble condition¹⁴, apparently owing principally to the formation of α -tricalcium phosphate¹⁵. In this connection it is interesting to note that with the Florida pebble phosphate increasingly close correlation between the citrate-solubility of the phosphorus and the volatilization of the F'_B fluorine was obtained by progressively increasing the thickness of the charge to an extent corresponding approximately to that of a four-grain layer of particles. It seems, therefore, that with this type of rock the optimum degree of contact between the silica and phosphate particles is not obtained in single-grain layers, whereas the optimum degree of contact is obtained in single-grain layers of Tennessee brown-rock phosphate. However, other factors may be involved, because under nearly all experimental conditions Tennessee brown rock seems to be converted into the citrate-soluble condition more easily than is Florida pebble.

With all types of phosphate rock, no increase in the citrate-solubility of the phosphorus occurs until complete volatilization is effected of all the

fluorine in excess (F_A) of that required for formation of fluorapatite with the phosphorus plus that corresponding to the first atom (F_B) of fluorine in the fluorapatite equivalent of the phosphorus. From that point, however, the citrate-solubility of the phosphorus increases with increase in fluorine volatilization, not in direct proportion to the total fluorine volatilized but in proportion to the percentage of the second atom (F'_B) of fluorine volatilized, provided the temperature is in excess of 1350°C ., the rock contains sufficient silica (about 4 per cent or more) in intimate contact with the phosphate particles, and the product is cooled quickly. The relation between fluorine volatilization and phosphorus solubility under these conditions is shown in Fig. 1. In all these experiments the thickness of the charge was several times that of a single particle.

The solubility of the phosphorus usually does not agree closely with the volatilization of the F'_B fluorine when: (1) silica-rich phosphate (Florida pebble, for example) is heated at 1350°C . and lower temperatures; (2) rock containing less than about 2 per cent of silica is heated at temperatures up to at least 1400°C .; (3) certain types of silica-rich phosphate (Florida pebble, for example) are heated in single-grain layers at temperatures up to at least 1400°C .; or (4) the product is cooled slowly¹⁰. In general, volatilization of fluorine in a quantity less than that corresponding to F_A plus F_B decreases the citrate-solubility of the phosphorus in the product below that of the phosphorus in the untreated rock.

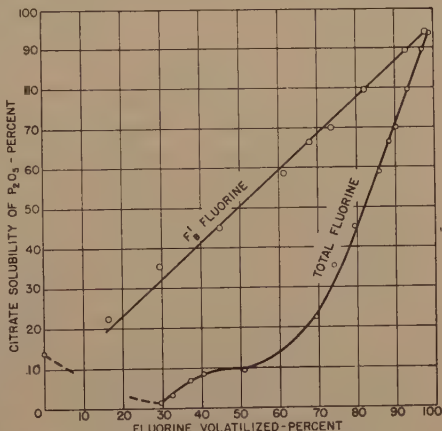


FIG. 1.—RELATION BETWEEN FLUORINE VOLATILIZATION AND CITRATE-SOLUBILITY OF PHOSPHORUS IN FLORIDA LAND-PEBBLE PHOSPHATE HEATED AT 1400° TO 1425°C .

FERTILIZER EFFICIENCY AND CHARACTERISTICS OF CALCINED PHOSPHATE

The fertilizer efficiency of calcined phosphate has been investigated under greenhouse conditions by Jacob and co-workers¹⁶. The results, which are summarized in Table 5, show that removal of only 50 per cent of the total fluorine (corresponding to approximately 80 per cent of the F_A plus F_B fluorine in the original Florida pebble phosphate) from calcined phosphates caused a reduction in the citrate-solubility and a marked decrease in the relative nutrient value of the phosphorus, as compared with that of the phosphorus in the original phosphate rock. With further removal of fluorine there was, in general, a progressive increase in the citrate-solubility and the relative nutrient value of the phosphorus. Calcined phosphates (samples 7 to 9) from which 90 per

cent or more of the total fluorine had been volatilized (citrate-solubility of 65 per cent or more) showed higher average relative nutrient values than did superphosphate.

TABLE 5.—*Fertilizer Efficiency of Calcined Phosphate*

Sample No.	Phosphate	Average Relative Efficiency of P_2O_5 as Indicated by Increase in:		Availability of $P_2O_5^d$, Per Cent	Fluorine Volatilized in Preparation of Calcined Phosphate, Per Cent	
		Dry Weight of Plants ^c	P_2O_5 Content of Plants ^c		Total	F'B
1	Superphosphate.....	100.0	100.0	99.6		
2	Dicalcium phosphate....	110.6	111.4	100.0		
3	Florida pebble-phosphate rock, untreated material	66.6	67.1	13.2		
4	Calcined phosphate ^a	19.7	15.6	6.8	49.9	0
5	Calcined phosphate ^a	54.9	44.5	20.6	64.2	1.6
6	Calcined phosphate ^a	85.9	75.2	49.1	78.9	44.0
7	Calcined phosphate ^b	109.4	106.6	65.4	90.0	76.0
8	Calcined phosphate ^a	109.4	104.5	78.1	93.3	82.2
9	Calcined phosphate ^b	116.2	125.9	86.1	97.0	93.0

^a Prepared from Florida pebble phosphate.

^b Prepared from Tennessee brown-rock phosphate.

^c Based on the increase from superphosphate as 100; average results obtained in six series of experiments.

^d Determined by neutral ammonium citrate method.

Characteristics.—The product is obtained in the form of a sintered or semifused clinker which, unlike superphosphate, requires no aging and needs only to be ground to the desired fineness for fertilizer purposes. Calcined phosphate is practically insoluble in pure water but is readily soluble in ammonium citrate and dilute citric acid solutions. It is weakly alkaline in reaction, has no deleterious effect on fertilizer bags and machinery, and should prevent, to a considerable extent, increase in soil acidity caused by the use of ammonium salts as fertilizers. In fertilizer mixtures, the alkalinity of the material is sufficient to cause some loss of ammonia from ammonium salts. Calcined phosphate absorbs only a small quantity of water in highly humid atmospheres, and the mechanical condition of the ground material (unmixed with other substances) is not altered by storage under adverse temperature and humidity conditions. The properly prepared material should contain about 30 per cent or more of available phosphoric oxide (P_2O_5), as compared with about 19 to 20 per cent in the best grades of ordinary superphosphate. As shown by greenhouse pot tests, the plant-food value of the phosphorus in the properly prepared material is as high as that of the phosphorus in superphosphate and dicalcium phosphate. Properly prepared calcined

phosphate is low in fluorine and offers possibilities as a substitute for bone meal in the preparation of mineral feeds for livestock. In many of its properties, calcined phosphate is similar to high-grade basic slag, which is an important phosphate fertilizer in European countries.

TECHNICAL AND ECONOMIC CONSIDERATIONS

The data presented in this paper and elsewhere^{9,10,14} show that intimate contact of the phosphate particles with water vapor and silica is necessary for efficient volatilization of the fluorine and conversion of the phosphorus into the citrate-soluble condition. It would seem, therefore, that any type of apparatus in which these conditions can be maintained at a sufficiently high temperature might be suitable for the process.

The preparation of calcined phosphate is similar to that of Portland cement in the respect that at the minimum temperature (about 1400° C. for both materials) for satisfactory reaction the product is obtained in a sintered or semifused condition. Calcined phosphate has been prepared experimentally in internally fired rotary kilns under operating conditions similar to those prevailing in the manufacture of cement. Such kilns are probably suitable for the commercial manufacture of the material at a temperature of 1400° C. It is probable that the reaction can also be carried out at temperatures above the melting point of phosphate rock. However, this would require a special method for introducing water vapor into, or contacting it with, the molten rock.

Although the authors are not in possession of sufficient data to justify an estimate of the probable cost of producing calcined phosphate on a large scale, the similarity of the process to cement manufacture indicates that the material can be produced, per unit of available phosphorus, at a figure lower than the present cost of superphosphate.

The fertilizer manufacturers of this country have millions of dollars invested in superphosphate and sulfuric acid plants. Furthermore, the fertilizer industry consumes annually a considerable tonnage of byproduct sulfuric acid, the greater portion of which is produced in connection with metallurgical operations. It is estimated¹⁷ that in 1934 the fertilizer industry used 1,450,000 short tons of sulfuric acid (basis 50° Bé.) or 25.6 per cent of the total domestic consumption. A reduction in the manufacture of superphosphate will result in a corresponding decrease in the consumption of sulfuric acid. If calcined phosphate or some other type of phosphate fertilizer should offer the possibility of serious competition with superphosphate, a strenuous effort would undoubtedly be made to reduce the cost of superphosphate.

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DISCUSSION

(R. W. Smith presiding)

R. W. SMITH,* Atlanta, Ga.—Is there any possibility of fluorine that is driven off in this calcination process being of commercial value, say for insecticides? Is the

* Acting State Geologist.

phosphate that is thus treated in the calcination process suitable for introduction in animal feeds? That was tried with the ordinary phosphate rock, with rather disastrous results because of the fluorine.

K. D. JACOB.—The question of the value of the fluorine that is driven off is rather hard to answer at the present time. In the manufacture of superphosphate approximately 25 per cent of the fluorine in the phosphate rock is volatilized, but because of the comparatively small demand for fluorine compounds only a few plants in this country are recovering the volatilized fluorine. Experiments have not been made on the recovery of fluorine in connection with the production of calcined phosphate, and it is not known whether the fluorine can be recovered economically.

Until recently a rather considerable quantity of raw phosphate rock, which contains 3 to 4 per cent of fluorine, was used annually as a phosphatic supplement in the feeding of livestock, particularly in the Middle West. However, we now know that that is a very dangerous practice because of the harmful effects of fluorine on the bones and the general health of animals.

If calcined phosphate comes on the market it will undoubtedly have a place in the mineral feed industry. In order to obtain the maximum fertilizing value of the phosphorus in calcined phosphate, the product should contain less than 0.1 per cent of fluorine. Bone meal, which is a safe and widely used source of phosphate for animal feeds, usually contains less than 0.1 per cent fluorine.

H. RIES,* Ithaca, N. Y.—Dr. Jacob made reference to a Russian apatite. Is much of that being imported into the United States and what is it being used for?

K. D. JACOB.—About 7000 tons was imported at Baltimore in the fall of 1932. The Russian apatite, as mined, is chiefly low-grade material and the greater part of the production is concentrated by a flotation process. The producers of phosphate rock in the United States protested the importation of Russian apatite on the ground that the product was concentrated by a flotation process owned and patented in the United States, which was used by the Russians without the payment of royalties or the consent of the owners of the patents.

After a hearing on the subject, the Tariff Commission decided that importation of floated Russian apatite constituted unfair competition with the domestic phosphate rock industry. However, the decision of the Tariff Commission was reversed by the Court of Customs and Patent Appeals, and shipments of Russian apatite were received in the Baltimore, New York and Rhode Island customs districts in 1935. The importations of Russian apatite were used in the manufacture of superphosphate and high-grade phosphate chemicals, principally the former.

G. R. MANSFIELD,† Washington, D. C.—Will this process make it advantageous to use lower grades of phosphate rock instead of higher grades, or perhaps supplementing the higher grades that are now on the market? In regard to the content of fluorine, is there any risk, if the process should be developed in a large way, of a repetition of the difficulty with the agriculturalists on account of the possible deleterious effect of fluorine in the atmosphere?

K. D. JACOB.—We have not obtained satisfactory results with low-grade rock in our laboratory experiments, apparently because the low-grade rock tends to fuse more or less completely at 1400° C., which is the minimum temperature for rapid reaction, and efficient contact between the phosphate and water vapor is prevented thereby. On the basis of our present knowledge, it seems desirable to use the customary com-

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mercial grades of phosphate rock for the manufacture of calcined phosphate. It will probably be possible to use low-grade rock if an economic method of obtaining efficient contact between the water vapor and fused rock can be worked out. Of course, the grade of the finished product will depend upon the grade of rock used. As to the second question, there is no doubt that provision will have to be made for the removal of fluorine from the furnace gases before they are discharged to the atmosphere.

MEMBER.—What is the grade of the Russian concentrate—the apatite concentrate?

K. D. JACOB.—The apatite concentrate is a very high-grade material. It usually contains 38.5 to 39.5 per cent of P_2O_5 . Also, a crude, unfloated material, which is marketed as lump apatite and contains 27.5 to 32.0 per cent of P_2O_5 , is produced.

P. M. TYLER,* Washington D. C.—Talking about Russian apatite, could you give us an idea of how long or how much longer is required to take the superphosphate out of the Russian apatite than out of our phosphate rock? I understand in Europe they are using that in the steel industry, but of course there it does not make so much difference, because of the nature of the material.

K. D. JACOB.—I understand that the Russian apatite is not entirely satisfactory for the manufacture of superphosphate by the processes used in the United States. The apatite is low in carbon dioxide. I understand that the superphosphate lacks the spongy structure that is characteristic of superphosphate made from phosphate rock and that it has a "dead" appearance and does not "cure" as rapidly as does normal superphosphate. I am told by a gentleman connected with a Swedish superphosphate company that they prefer to mix the apatite with phosphate rock for the manufacture of superphosphate, although they have found that a satisfactory superphosphate can be made from Russian apatite alone if certain changes are made in the manufacturing process.

W. T. SCHALLER,† Washington D. C.—The development of apatite on the Kola Peninsula has caused the formation of a city, according to the reports of the Russian Government, of 40,000 inhabitants, where a few years ago there was nothing but a wild waste.

K. D. JACOB.—That is true. The bulk of the exports of apatite is shipped from the port of Murmansk, which is the only port in the northern part of the Soviet Union that is free from ice throughout the year.

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American Potash Mines Prepared to Produce Over a Million Tons of Crude Salts Annually

BY HOWARD I. SMITH,* MEMBER A.I.M.E.

(New York Meeting, February, 1935)

At the meeting of this Institute in February 1933, I presented a paper on potash development in southeastern New Mexico¹, which contained a brief review of the Geological Survey's 20-year search for commercial sources of soluble potash minerals and of the events that led to the discovery, exploration, and first development of the deposits in that state. As a more comprehensive review of the subject was published in December², the present paper is intended only to discuss recent developments in the domestic potash industry, particularly in New Mexico. A summary of the findings of potash in the Permian Basin, by G. R. Mansfield and W. B. Lang, geologists of the United States Geological Survey, is now in process of publication by the University of Texas and will be omitted from this paper.

In New Mexico prospecting for potassium salts during the past two years has increased materially the known area of commercial deposits; development has resulted in an increase from one to three in the number of shafts and the starting of a fourth; an increase of annual productive capacity to 1,000,000 tons of crude salts, or 250,000 tons of K_2O ; and an increase in annual refining capacity to 140,000 tons of muriate of 62.5 per cent grade. Construction has been completed to increase still further the production of high-grade salts.

In California the American Potash & Chemical Co., operating on Searles Lake brines, has increased its plant capacity from about 100,000 tons to 180,000 tons of muriate a year and is now producing soda ash and salt cake in addition to potash and borax.

Potash production from cement-plant and industrial-alcohol wastes remains unimportant.

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¹ H. I. Smith: Potash Development in Southeastern New Mexico. A.I.M.E. *Contrib.* 52 (1933).

² H. I. Smith: Three and a Quarter Centuries of the Potash Industry in America. *Eng. & Min. Jnl.* (1933) 136.

Imports of potash salts from Spain, Russia, and the Dead Sea region have been added to those from Germany, France and Poland. This keen competition for the American potash market, a market second only to that of Germany, has resulted in the cutting of prices to levels reported as low as one-half those quoted in 1932. In consequence the growth of the domestic industry has been somewhat retarded, and, instead of contesting in world production with Poland for third place among the potash-producing countries, the United States may be forced to yield both third and fourth places to Russia and Spain respectively, by reason of its inability to duplicate the low labor costs in those countries, its own unrestricted admission of potash imports free of duty, and its exclusion from the principal foreign markets by the duties, quotas, or embargoes involved. The present outlook is that American producers will have captured less than 50 per cent of the American market in 1934. Poland, which ranked third in 1932, was relegated to sixth place in 1933.

THE NEW ERA IN AMERICAN POTASH PRODUCTION

The new era in American potash production, begun in 1932, was based, as stated, on more than 20 years of preparation initiated in consequence of unsatisfactory trade relations, which induced in Congress a disposition to foster the search for a competent source of domestic supply. The potash famine during the World War, and the activities of the Geological Survey, greatly stimulated both public and industrial interest in finding mineral potash.

The intensive search for a domestic source of potash from 1910 to 1925 by the different Government agencies included explorations of all known potential sources of potassium brines, minerals and organic substances. The output of potassium salts during this time was of great national importance but the investments therein were in general financial failures. Beginning in 1925 with the discovery of sylvite (potassium chloride) in New Mexico and subsequent drilling and discovery of sylvite by the Government and six other agencies or individuals, the development of the discovered deposits by two companies, a new mining industry was definitely established in America and independence from foreign sources for potash salts now definitely established.

Today about 1500 men are employed in the production of potash, principally in California and New Mexico. Taxes and royalty charged against the industry in New Mexico during 1934 alone are estimated at \$150,000, including a special tax levied in 1934 by the state legislature, consisting of 2 per cent of the value of crude salts sold and of 0.5 per cent of the value of refined salts sold. Statistics showing the trend of the industry are well presented in the Minerals Yearbook for 1934, published by the United States Bureau of Mines.

RECENT DEVELOPMENTS IN NEW MEXICO

During the last two years the capacity to produce potash salts in New Mexico has been increased to more than 1,000,000 tons of crude salts annually.

The United States Potash Co., which completed its first shaft (for hoisting) in 1931, completed its second shaft (for air) in 1933, at a depth of 999 ft. 7 in. The second shaft is about 2000 ft. south of the first. When the two shafts were connected by the underground workings in June 1933, restrictions imposed by the Federal Mining regulations on the number of men employed underground at any one time were removed and freedom of production was enjoyed. The present mine capacity of more than 100 tons an hour is limited only by the capacity of the grinding plant at the mine. The refinery has a daily capacity of 400 tons of refined muriate. This company is now prepared to supply the trade with any grade of product desired, from manure salts with the equivalent of 25 per cent K_2O to muriate of a purity acceptable to the chemical trade. The mine is equipped with 24-in. gage tracks and side-dump cars. The salt produced at the face is loaded mechanically, dumped into the ore bin at the shaft bottom, hoisted to the surface in 4.6-ton skips, and there ground to a size suitable for shipment either direct to the trade as manure salts or to the refinery. Water for the refinery is obtained from the Pecos River, and the part used for condensing is available thereafter for irrigation or can be returned to the river. The part used in the process of refining is later used to flush refinery residues to a convenient playa, where the water is allowed to evaporate. Rights to the water were acquired by the purchase of agricultural lands carrying such rights. Natural gas is used for a fuel for the development of the steam required for refining and for the generation of electricity.

The Potash Company of America started¹ sinking its first shaft in February 1933, and completed it in December of that year. The depth to the top of the rail is 993 ft. Later the shaft was sunk to about 1100 ft., to provide bins for skip loading. The temporary headframe and hoist installed for sinking are still in use. A second shaft was started Oct. 27, 1934, about 700 ft. northwest of the first. The salt, when mined, is loaded into 4-ton cars on 42-in. gage track and later will be dumped by a rotary dump into an ore pocket at the shaft bottom, then hoisted by skips of 2-tons capacity. The salt is then ground to 8 or 10 mesh for shipment direct as manure salts. When the ore-dressing equipment is installed in a near-by building the ground salt will be used for direct shipment or conveyed to the concentrating building where the halite is to be eliminated by ore-dressing methods to produce the grades of salt normally required by the fertilizer trade. The separation of halite and sylvite, which differ only 0.16 in specific gravity, is an important develop-

ment worked out under a cooperative agreement between the company and the United States Bureau of Mines.* The plant is expected to be in operation during the early summer. The coarse-grinding plant of this company has a capacity of 150 tons an hour. It is now at the head of the shaft, but the plans call for placing it underground. The first unit of the grinding plant now in operation has a capacity of 50 tons an hour. Production, however, is limited by the number of men permitted to work underground at one time until the second shaft is completed and connections are made to provide a second escapeway, in conformity with Federal regulations.



FIG. 1.—AIRVIEW OF SURFACE PLANT OF POTASH COMPANY OF AMERICA ON FEB. 11 1935, ONE YEAR AFTER THE SHAFT WAS STARTED. COURTESY OF THE COMPANY.

The names of many other potash companies or associations interested in potash have appeared in the New Mexico papers. None of them have progressed beyond the stage of prospect drilling, making investigations, and formulating plans. Those that have been the most active are the New Mexico Potash & Chemical Co., the Independent Potash & Chemical Co., Kroenlein and associates, and the Texas Potash Corporation of Colorado, which recently has been reorganized as the General Potash Co. to avoid confusion with another organization of the same name in Texas.

Outside New Mexico progress in the potash industry has been restricted to plant enlargement and betterments by the American Potash & Chemical Co. at Searles Lake, Calif. Little change has taken place in the recovery of potash salts from molasses waste and cement dust beyond the development of a process for the recovery of fine dust particles, which as early as 1917 were recognized as containing less calcined raw material

* See page 341.

and more alkali salts than the coarse particles³. In the new process the coarse material is eliminated by the use of multiclones in series, and the fine dust particles, usually less than one micron in diameter, are collected by the use of an electrical precipitator. It was anticipated that, with the expiration of the A. C. Spencer patents and the improvements made, the recovery of potash from this source would quickly increase, but to date no appreciable increase in production has resulted. The fine particles yield a product containing the equivalent of about 22 per cent K_2O , primarily as sulfate, compared with 10 per cent from the unclassified dust. The 22 per cent product is directly marketable, whereas the crude material, with 10 per cent or less, necessarily required selling in local markets or leaching and precipitation before shipment, in order to avoid excessive freight charges.

No significant discoveries of potash have been made recently in the United States outside the Carlsbad area of New Mexico, although confirmation has been provided of a previous discovery of sylvite in eastern Utah, and small quantities of mineral potash have been observed in the salt beds of Kansas.

PROSPECTING SINCE 1932

The New Mexico potash field is in the western part of the Permian Basin, in which deposits have not yet been found to extend appreciably west of the Pecos River. The deposits are in an area having no well defined surface drainage. The plateau country to the east is called the Llano Estacado (staked plain), from the use of stakes by the early explorers to mark their trails across an area devoid of natural landmarks. The surface water in the thousands of square miles of this plateau is generally confined to small lakes or sinks. Subsurface water, however, is found at various depths in most drill holes and locally only a few feet below the surface. Six different water horizons above the top of the salt have been correlated by R. V. Ageton, of the Geological Survey.

The part of the Permian Basin most intensively prospected by core tests is an area of about 100 square miles lying some 20 miles east of Carlsbad. Of the 87 core tests drilled in the Permian Basin, the results of which are available to the Geological Survey, 60 are in this area. Here the occurrence of sylvite in deposits or beds 4 to 12 ft. thick containing an equivalent of 14 to over 37 per cent K_2O has been established in an area of about 35 square miles, within which four subordinate areas have been outlined containing beds with an equivalent of 25 per cent or more of K_2O . Two of these areas of relatively high-grade salts are now under development. Another is covered by prospecting permits, and the fourth is in a Government reserve subject to such outstanding rights as may be sustained.

³ H. S. Gale: *Mineral Resources U.S. for 1916* (1919) pt. 2, 134.

During the past two years the United States Potash Co. has increased then umber of its core tests from 27 to 28, the Potash Company of America from 15 to 17, and the New Mexico Potash Co. from 2 to 6. Of the newcomers, the General Potash Co. has drilled 6 holes and Kroenlein and associates 1. These, with 4 drilled by the Gypsy Oil Co. and 3 by the Federal Government, aggregate 65 core tests, all but 5 of which are within the 100 square miles constituting the present New Mexico field. No difficulty has thus far been experienced in correlating the logs and cores, the same sequence of certain beds apparently prevailing within



FIG. 2.—SURFACE PLANT OF MINE OF UNITED STATES POTASH CO. IN 1934. COURTESY OF THE COMPANY

the areas intensively prospected, regardless of a limestone or coral reef that is thought to extend across the area and was at first presumed to limit the occurrence of exploitable deposits. Correlation of beds, however, does not imply continuous potash tenor, which in fact is found to change radically within short distances.

The reserves have not been recalculated since the end of 1932, when three subordinate areas of high-grade salts were conservatively estimated to contain more than 100,000,000 tons⁴ of crude salts containing over 14 per cent K_2O . Drilling during the last two years both north and south of the properties of the United States Potash Co. has disclosed evidence warranting an increase in the estimates of known reserves. Langbeinite, an anhydrous potassium-magnesium sulfate containing, when pure, 22.7 per cent of K_2O , has been found in cores by five different interests in the New Mexico field. After the percentage of langbeinite in

⁴ H. I. Smith: Reference of footnote 1.

these cores has been determined its economic importance can better be appraised.

Little change has been made in the technique of prospecting. Double core barrels designed for work in the salt area are used exclusively. All core drilling has been done by the Pennsylvania Drilling Co. and the Sullivan Machinery Co. with the exception of one test, drilled dry, by D. C. Muncy. Most of the drilling is done by contract at a definite price per foot of core recovered and a supplemental price for depth.

In nearly all the tests 2½-in. cores are now being obtained in the salt section without etching of the sylvite. The latest method of control in maintaining a properly saturated drilling solution is testing for a precip-



FIG. 3.—REFINERY OF UNITED STATES POTASH CO., 1934. CAPACITY OF REFINERY 140,000 TONS ANNUALLY. COURTESY OF THE COMPANY.

itate with a 1° drop in temperature. If no precipitate occurs the proper salts are added to make the solution saturated at bottom-hole temperature. In the core test drilled on the Crosby permit by Mr. Muncy no liquids were used. The core was cut dry, and the cuttings were blown from the hole with compressed air. This method appears to have some advantage in certain formations that tend to swell or run in the presence of liquids. The methods of analysis of potash samples vary: One company uses the platonic chloride method; others use a modified cobaltic nitrite method. Both methods are reported to be satisfactory when correctly used.

POTASSIUM SULFATE FROM DOMESTIC SOURCES

At present essentially all the potassium sulfate and potassium-magnesium sulfate consumed in the United States is imported.

Since the Bureau of Mines presented a report of progress on its potash investigation before this Institute two years ago, two new process outlines have been developed by that Bureau and a number of process steps have been tested out in small-scale engineering equipment.

During the past two years two proposals to produce potassium sulfate from domestic sources have been receiving public attention, one by the

Bohn Aluminum & Brass Corporation involving Utah alunite and another by the New Mexico Potash & Chemical Co. and the General Potash Co. involving the production of langbeinite from the bedded saline deposits of New Mexico. Other possibilities have been proposed. Economical methods for the recovery of potash from alunite have long been sought. The recovery of langbeinite from a mixture with sylvite and halite appears to offer little difficulty and may be of economic interest where the deposits contain the equivalent of not less than 25 per cent K_2O and are found in



FIG. 4.—REFINERY AND PLANT OF AMERICAN POTASH & CHEMICAL CO., TRONA, CALIF. YEARLY CAPACITY 180,000 TONS OF POTASSIUM CHLORIDE. COURTESY OF THE COMPANY.

beds over 4 ft. thick. If the content of langbeinite is particularly high this percentage may be lowered.

Some of the domestic sulfate market could probably be satisfied by a partial substitution of American muriate, by reason of its relative purity and much lower chlorine-potassium ratio than that of ordinary manure salts. The maximum allowable for some crops is 2 per cent of chlorine in the mixed fertilizer. However, a certain amount of chlorine is often desirable. A mixture of crude potassium and magnesium sulfates, such as langbeinite, enriched with high-grade muriate, should meet with favor in the fertilizer trade—first, because of the satisfactory ratio between potassium-magnesium and chlorine; second, because of the slow solubility of langbeinite, which extends its availability over a longer growing season; and third, because of the richness of the mixture and its resistance to caking as contrasted with imported potassium-magnesium fertilizers. Langbeinite is the principal source of the potassium sulfate produced in Poland.

FOREIGN DEVELOPMENT

Foreign development of potash since 1932 has been as rapid and unexpected as in the United States, and although France and Germany still are the principal sellers to America, Spain and Russia have been successful in capturing a remarkable proportion of the American trade.

The mine at Suria, Spain, with its 10 per cent salts, has received little consideration as a possible contender for the American market. A mine opened at Cardona in 1930, however, produced in 1933 potash averaging 15 per cent of K_2O . In 1932 the Ibericus mine, near Sallent, was opened, and in 1933 its salts averaged 18.24 per cent. In 1933 the output at Suria declined 18.1 per cent, whereas that at Cardona increased 24.4 per cent and that of the Ibericus mine 812.6 per cent. It is entirely natural that when salts of higher grade than these being worked are discovered they should be developed, even though there is not sufficient market to maintain the output. The mine at Suria with its low-grade salts has been affected more than the others. Exports in 1933 took 85 per cent of Spain's production. The Spanish deposits are only about 50 to 60 miles from Barcelona and thus have a much shorter haul from the mines to seaboard than those of any other country. The mine at Suria, with its cartel connection, is in a position to supply a part of the cartel market, which it now badly needs to maintain production.

Little definite information is available concerning the widely heralded potash deposits of the Soviet Union (Russia), although they have been declared to be the richest and most extensive in the world. The first mine to produce is at Solikamsk. From statements published in *Pravda*, July 12, 1934, to the effect that 3500 tons of ore is refined daily at this mine and that the refinery produces 600 to 650 tons of 85 per cent muriate daily, it is inferred that the mine-run salts average about 10 per cent K_2O . Another mine is being put into operation at Berezniki. The *Vossische Zeitung* (Feb. 22, 1934) reports that this new shaft can technically deliver 20 per cent K_2O .

Potash imports from Russia have been sold in the American market substantially below domestic prices, with an indication that price is a secondary consideration to the Russian producers.

The Franco-German cartel has countered the Spanish, Russian and American producers in many ways, including reductions of price to meet Spanish competition, and warehousing in America and adjustments of price to future market declines in such manner as to forestall shipments of American producers during the fertilizer season. The American producer has been largely eliminated from the manure-salt market by reason of these guaranties as to price decline and the high freight charges from Carlsbad, N. M., to the principal points of consumption. Indications are that the total production of crude salts in 1934 in New Mexico will

be less than in 1933, even though the capacity to produce is greater. Present prices are about 50 per cent below normal and are about the same as those stipulated in the contracts of American concerns with certain German producers in 1910. They are probably the prices at which the most favored mines in Germany can produce and sell potash salts at a profit, though doubtless they are below the average price at which German potash can be sold at a profit if interest and amortization of bonded indebtedness and investment in idle mines are considered. The price to the German consumer was reduced between 11 and 38 per cent late in 1934, indicating a tendency toward lower world prices.

POTASH NEEDS

Even in Revolutionary times the soils of the East were found to require fertilizing, for Washington is said⁵ to have recorded that wheat planted on exhausted tobacco land yielded but 8 to 10 bushels to the acre. He felt that future generations were being despoiled by the destruction of lands that the owners occupied merely for life, and interested two English agricultural experts to visit him at Mount Vernon.

The present need for potash is probably greater for quality of agricultural products obtained by its use than for quantity, as was abundantly illustrated during the period of potash scarcity at the time of the World War. Summaries of fertilizer tests too frequently appraise potash utility in terms of quantity of resulting product and too rarely in terms of quality.

The potash annually removed from the soil through cropping alone is equal many times over to the amount artificially applied. Other quantities are removed by leaching or are made unavailable to plants by fixation in the soil materials. Even the amount exported in wheat and other agricultural products is greater than that applied to the soil. The addition of larger and larger quantities to the soil appears necessary in the future.

When we depended upon imported potash salts and domestic phosphate for fertilizers the natural tendency was to minimize potash and emphasize phosphate. Now with domestic production of both it is natural to assume that there will be some readjustment in favor of potash. The need of research and the dissemination of information leading to more extended use of American potash is very apparent.

CONCLUSIONS

Prior to 1932 many of the mines in Germany were closed or abandoned in consequence of an unwise governmental policy that permitted excessive developments in that country. The new mines developed in Spain,

⁵ Union Pacific Coal Co. *Employees Magazine* (February, 1934) 51.

Russia, and the United States since 1932 have so affected the world market that the potash deposits in every producing country can almost be regarded as overdeveloped. However, with reference to national protection, a single source of potash is scarcely considered adequate. With two sources in Russia, three in Spain with two others under development, three in Poland, one in Palestine, three in the United States, and a large surplus in France and Germany, the world appears well supplied, and it is only natural that the prices established by the European cartel should respond to production over which it exercises no control.

The struggle for world markets has begun with a cutting of prices and dumping, accompanied by restriction of imports into the several producing countries except the United States, where all countries are free to sell as much potash as their agents can market. In the United States, with two mines in New Mexico and one plant in California, development is in a fairly healthy condition, but precaution is necessary, possibly through governmental influence, to prevent overdevelopment, such as has occurred in Germany and to a lesser extent in France and Spain. The potash produced in Poland and the United States is intended primarily for domestic consumption.⁶ The United States producers are meeting low prices at home without national protection but are eliminated from the world markets that are open only to agencies able to supply both sulfate and muriate salts. As soon as American operators start the production of sulfates they will be in a more advantageous position in both domestic and foreign markets.

DISCUSSION

(W. M. Weigel presiding)

E. P. PARTRIDGE,* New Brunswick, N. J. (written discussion).—The rapid and healthy development of a new potash industry in New Mexico to supplement that previously established in California should be a subject for mutual congratulation, tempered by the caution proper to engineers. The domestic producers of potash salts must face not only the immediate competition from groups in other nations operating under the profit system, but also the possibly more important future competition of Soviet plants operating under a nonprofit system. The domestic producers must, moreover, meet this competition without relying to any great degree upon tariff protection, unless they possess unusually potent ability to persuade the political representatives of the ultimate potash consumers.

Under these conditions particular interest centers upon possible technical developments that may lower the cost of production or offer special potash materials to supplement the chloride.

With respect to the reduction of costs in the manufacture of potassium chloride, the mechanical separation of sylvite from halite by tabling, recently worked out by

⁶H. I. Smith: Potash Mined in the United States May Meet All Domestic Needs. *Oil, Paint and Drug Reporter* (June 29, 1931).

*Supervising Engineer, Nonmetallic Minerals Experiment Station, U. S. Bureau of Mines, Rutgers University.

the U. S. Bureau of Mines in cooperation with the Potash Company of America, may prove to be a most important development if full-scale operation substantiates experimental tests. More authoritative information concerning this process will be offered by other discussers.

With respect to the production of potash salts other than the chloride, some mention may be made of the potentialities of polyhalite. Because sylvite, a mechanical mixture of potassium chloride and sodium chloride, can be refined by a simple and well-known process, the discovery and exploitation of rich deposits of this material in New Mexico has caused some casual observers to regard polyhalite as more or less a poor relation. Polyhalite, however, appears to be a logical source for the not inconsiderable potassium sulphate requirements of the United States.

In contrast to sylvite, polyhalite is a complex salt comprising potassium sulphate, magnesium sulphate, and calcium sulphate chemically bound together with water of hydration. The recovery of potassium sulphate from polyhalite is more involved than the recovery of potassium chloride from sylvite, but the higher unit value of the sulphate and the value of potential by-products such as magnesia and sodium sulphate also must be considered.

During the period of somewhat more than six years since the core-drilling investigation of the U. S. Bureau of Mines and the U. S. Geological Survey revealed the widespread occurrence of polyhalite in western Texas and New Mexico, the Nonmetallic Minerals Experiment Station of the Bureau of Mines has studied methods for recovering potassium salts and other industrial chemicals from this mineral. Two years ago the various methods of treatment developed up to that time were described before this Institute. Since then the fundamental operations of washing out sodium chloride, calcining, extracting with hot water, and producing syngenite have been proved by tests on a semiplant scale; the study of the reduction of polyhalite with natural gas to produce potassium sulphide has been continued; and two new process outlines for the recovery of potassium sulphate have been worked out in the research laboratory.

The first of these new processes, which involves the conversion of polyhalite to a syngenite-gypsum mixture and the two-stage extraction of potassium sulphate from this mixture by autoclaving at 200° C., is estimated to give a recovery of 87 per cent of technically pure potassium sulphate, and, as by-products, a solid residue of calcium sulphate, and a concentrated magnesium sulphate liquor suitable for the manufacture of sodium sulphate and magnesia.

The second new process was developed to utilize the sulphate content of polyhalite for the manufacture of potassium sulphate from potassium chloride. It involves the conversion of polyhalite into a syngenite-gypsum mixture, decomposition of this by means of ammonia and carbon dioxide to produce a liquor containing ammonium sulphate and potassium sulphate, addition of potassium chloride equivalent to the ammonium sulphate, and precipitation of potassium sulphate by ammonia. This process, which would consume approximately equal quantities of crude polyhalite and crude sylvite of the grades found together in New Mexico, has an estimated recovery of 86 per cent of the entering potassium as technically pure potassium sulphate. A concentrated magnesium sulphate liquor and a concentrated calcium chloride liquor would be obtained as by-products.

Because of the severe curtailment in its appropriations combined with the pressure of other problems in the nonmetallics field, the Bureau of Mines has found it necessary to discontinue its polyhalite investigations. Although much valuable work might still be done, it is believed that a sound basis for the development of a potash industry utilizing polyhalite has already been laid. If the domestic production of potassium sulphate is as desirable as Mr. Smith has indicated, an early renewal of active commercial interest in the possibilities of polyhalite may be anticipated.

R. S. DEAN summarized *Report of Investigations* 3271, U. S. Bureau of Mines, written by Will H. Coghill, F. D. DeVaney, J. Bruce Clemmer, S. R. B. Cooke:

The usual chemical method for recovery of potash from sylvite, while efficient in production of pure potassium chloride, requires expensive equipment and is costly to operate. It occurred to the Bureau of Mines that ore-dressing methods that have been applied to other nonmetallic minerals might be adapted for the production of a grade of potash suitable for the fertilizer industry. After preliminary study by the Bureau of Mines showed that ore-dressing treatment was possible the investigation was extended under a cooperative agreement between the U. S. Bureau of Mines, the Potash Company of America, and the Missouri School of Mines and Metallurgy, Rolla, Mo.

Conclusions.—By maintaining a closed circuit of brine saturated with the constituents of the ore, the sylvite ores of Carlsbad, N. Mex., may be concentrated by ore-dressing methods. Any one of three procedures will yield high-grade concentrates and high recoveries: Table concentration supplemented by flotation, tabling and agglomerated feed supplemented by flotation, and all-flotation are effective. Plain gravity concentration on tables yielded concentrates assaying 82.7 per cent of KCl with a recovery of 69.8 per cent. When the fines were treated by flotation the final concentrates were 84.9 per cent KCl, and the recovery was 97.3 per cent KCl. The other two methods gave higher grade concentrates and recoveries equally as good.

H. M. ALBRIGHT,* New York, N. Y.—We now have ample plant capacity in California and New Mexico to supply all domestic needs for all grades of muriate of potash. The New Mexico mines have been developed to such an extent that they can more than meet the requirements of American agriculture for manure salts of various grades. In spite of this position, American potash producers supply less than half of the domestic requirements. High freight rates and high labor costs place us at a disadvantage. We pay miners \$5.85 per day; Spain pays a little over a dollar a day, France about \$1.35 and Germany around \$2. Foreign potash moves to American ports at low rates, from \$3 to \$5 a ton, and we understand that it is occasionally shipped as ballast; from New Mexico freight rates run from \$6 to \$12 a ton, depending on destination. We urge the inauguration of a policy that would assure the American producers a reasonable share of the domestic market. The potash industry is not comparable with the manganese industry; there is no reason whatever to encourage importation of potash.

MEMBER.—Is potash flotation now carried on commercially?

E. P. PARTRIDGE.—No, it is still in the laboratory stage.

L. D. ANDERSON,† Denver, Colo.—Mr. Smith has been most helpful in these developments of potash. Unfortunately, he appears to have been misinformed on one point; that is, the inference that the Potash Company of America is building a mill or refinery based upon the results of research by the Bureau of Mines. As a matter of fact, the process upon which the new works, now in course of construction, is based was developed after many months of research entirely independent of the Bureau of Mines and is not at all like the methods recommended by the Bureau.

H. I. SMITH.—Mr. Anderson's comments are well taken. However, at the time the paper was written the statement in the paper was correct and had the approval of the company. The Potash Company of America had been negotiating for equip-

* Vice President and General Manager, United States Potash Co.

† Consulting Engineer, Potash Company of America.

ment in harmony with tests conducted in cooperation with the Bureau of Mines and a building was built, or nearing completion, to house the equipment. After the paper was submitted additional research was made, primarily by Messrs. Anderson and Wein- ing, and the original plans were abandoned, the building was enlarged, and different equipment has been installed. The new plant is now expected to be in operation some time this month, or about a year after my paper was written. These later developments should normally be subject matter for another paper rather than for last minute revisions.

O. BOWLES,* Washington, D. C.—We should give credit to the pioneer companies and the research laboratories for recent spectacular developments in the domestic potash industry.

* Chief, Building Materials Section, U. S. Bureau of Mines.

Developments Affecting the American Potash Industry

BY HOWARD I. SMITH,† MEMBER A.I.M.E.

(New York Meeting, February, 1936)

For several years this Institute has recorded in its TRANSACTIONS the various discoveries of potash‡ in America, and the successive stages in the development of an independent domestic potash industry. This paper is intended primarily to continue the record through the year 1935. In retrospect it is not inappropriate to note that public interest aroused in 1910 resulted in the production of some 1090 tons of pure potash in 1915, as a small beginning in the domestic inorganic potash industry, and that the infant industry thus begun reaches its majority this year, following a record production in 1935 of about 600,000 tons of potash salts and exports to no less than 22 foreign countries, including six countries in Europe. This production, with an estimated 200,000 tons of pure potash, may be compared with a production in 1934 of 430,157 tons of salts with an equivalent of 144,342 tons of pure potash.

MINE AND PLANT IMPROVEMENT

During 1935, the American Potash & Chemical Co., at Searles Lake, Calif., began to reap the benefits of its previously enlarged plant facilities, which increased its productive capacity to 180,000 tons of muriate annually.

The United States Potash Co., at Carlsbad, N. Mex., improved its operations by installing trolley locomotives and other haulage facilities and is maintaining its development headings two years in advance of operations, with projected plans for orderly mining for years to come. The hoisting shaft at its mine is now equipped to hoist at the rate of 1,000,000 tons annually, and the refinery to produce 150,000 tons of muriate containing less than one per cent of impurities.

The Potash Co. of America, at Carlsbad, N. Mex., completed the conversion of its plant from a prospecting to an operating basis late in 1935. The coarse-crushing plant, with a capacity of 150 tons an hour, formerly on the surface, was installed underground, with ore pockets

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‡ The term "potash," when prefixed by "pure" or expressed in percentage, refers to K_2O equivalent; when unqualified, it refers to appropriate salts of potassium.

for skip hoisting to ore bins at the surface adjacent to the 125-ft. head-frame. The output and development of this company were restricted during 1935 because of the time consumed in converting the shaft and hoisting equipment for capacity operation and because of the low price and small profitable market for manure salts during the first half of the year. The air shaft, which was sunk to a depth of 12 ft. a year ago and stopped pending further underground exploration, was deepened to 245 ft. A water flow of about 1200 gal. per min. was struck and sinking operations again temporarily stopped. The plans call for the completion of the shaft by September, 1936. If it is completed, the Federal restrictions on the number of men permitted underground on one shift will then be removed, and by fall the mine will be prepared to hoist at the rate of 1,000,000 tons annually. The refinery at this mine has an output capacity of 200 tons daily. It is therefore evident that late in 1936 American mines and plants will have a capacity to produce primary salts at a rate in excess of 2,000,000 tons a year, from which about 400,000 tons of high-grade muriate can be made with present plant facilities. No new source of primary potash production is anticipated during 1936, although refinery capacity can readily be increased if demanded.

TECHNICAL ADVANCE

About 20 papers have been published by the engineers of the U. S. Bureau of Mines on experimental work at New Brunswick, N. J., on the production from polyhalite of potassium sulfate and other salts, and the final report on this work is in preparation. The most interesting phase of the work relates to the production of potassium sulfide by the reduction of polyhalite with natural gas and the possibility of converting the potassium sulfide thus obtained to the sulfate, carbonate and hydroxide of potassium. Other research work has been done privately on the production of sulfate from alunite, muriate and brines, and a moderate amount of sulfate has been produced.

Tests have been continued by the U. S. Bureau of Mines on the sustained bearing strength of specimens of salt from the mine of the United States Potash Co., and comparative observations and measurements are being made by company officials on mine pillars. This work, when concluded, will contribute materially to the maximum recovery of potash salts from the New Mexico operations.

The mechanical separation of sylvite and halite initiated by me and forecast in the report of the Bureau of Mines for the year ending June, 1934, was carried to a successful conclusion and turned over to the potash industry¹, and has been further developed by the Potash Co. of America. This most interesting and important development may be summarized as follows:

¹ References are at the end of the paper.

The Bureau of Mines, after making preliminary tests, stated in its annual report for 1934 that sylvite and halite are amenable to mechanical separation. Later that year, in cooperation with the Potash Co. of America, a more extensive program was undertaken. Preliminary to this cooperative work emphasis was placed on the desirability of producing a coarse-ground product, which was considered more acceptable to the market and in consequence small-scale agglomeration and tabling tests were undertaken at Rolla, Mo. After the completion of the preliminary tests, arrangements were made to conduct tests on a commercial scale at Golden, Colo. The results of these tests being favorable, a building to house the equipment was completed at the mine of the Potash Co. of America on Dec. 8, 1934, and when my previous paper² was written (1934), the company was negotiating for the necessary equipment to enter that season's market and concluded its cooperation with the Bureau of Mines. In the meantime the company started independent research on finely ground material. This further research work, done by A. J. Weinig, at Golden, Colo., with L. D. Anderson representing the company, was so promising that the purchase of the plant equipment was postponed for nearly a year, a new flow sheet was developed, the building was enlarged, flotation equipment was installed in October, and the first shipment of mechanically concentrated muriate was made on Nov. 9, 1935. This installation marks another stage of advancement in the so-called "ore-dressing field." The new plant, designed for 200 tons of concentrates daily, is reported to have prepared muriate with less than one per cent of impurities.

The Bureau of Mines successfully demonstrated that langbeinite could be separated from halite by flotation³. The prime advantage of concentration of potash salts by mechanical means lies in the reduced cost of plant and the reduced fuel and water requirements as compared with the usual refining process of dissolving and fractional crystallization from hot solutions. The full importance of the reduced water requirements can more readily be realized when one considers the difficulty of securing an adequate supply of water in the semiarid country where the deposits are located.

MAPS, ROADS AND EXPLORATIONS IN NEW MEXICO

Map.—The Geological Survey recently published in advance sheet form a topographic map of 349 square miles of the active potash area in Eddy County, N. Mex., east of the Pecos River, extending on the south from the refinery of the United States Potash Co. to and including the leased land of the Potash Co. of America. This new map, designated "Potash Mines Area," on the scale of 1 in. equals 4000 ft., with contour intervals of 5 and 10 ft., will fulfill a demand for a reliable map of the area.

Roads.—Access to the field has been improved by the relocated and reconstructed Carlsbad-Hobbs highway (U.S. 62) from Carlsbad, on the west, to a point east of the United States Potash Company's mine. A connecting road was graded and partly surfaced north from that highway to the mine of the Potash Co. of America. Other roads extend from Carlsbad to the refinery of the United States Potash Co. and to the holdings of the Union Potash & Chemical Co., a newly organized operating company on lands including the holdings of the Independent Potash & Chemical Co. and the Texas Potash Co. Other sections of the field are reached by semidesert trails difficult to follow but usually passable for automobiles.

Explorations.—Prospecting during the year was restricted to four core tests in New Mexico—the Parrott No. 1, Texas No. 7, and Harris Nos. 18 and 19 borings, all within a few miles of the producing mines. Additional geologic information concerning the field was furnished by cuttings from wells drilled for oil. According to Mansfield⁴, sylvite, langbeinite or carnallite was found in New Mexico in the cuttings from 18 holes drilled for oil or gas. He also gives the results of core-drilling by the Government.

Core holes, cuttings from oil and gas wells, and geophysical prospecting by the oil interests have provided the geologists an excellent fund of subsurface data on this field. As more information is obtained concerning the nature of the New Mexico potash deposits, the uncertainty of exploration by core-drilling becomes more evident. For instance, all the 40 potash beds so far discovered⁵ are independently irregular in shape and continuity. Bed 28 numbered from the top and now being mined in New Mexico, is interrupted in places by rock salt⁶. In drilling such beds a valuable deposit may be missed by a few feet, or if good salts are found at the horizon, rock salt may occur close by.

GOVERNMENT LAND

Prospecting permits issued by the Government authorize prospecting for potash on public land for a two-year period and are subject to extension for an additional term of two years on a satisfactory showing to the Secretary of the Interior of diligence during the initial term and of intent to continue prospecting on or for the benefit of such lands: By order of April 5, 1934, the Secretary of the Interior suspended action on all applications for potash-prospecting permits for a period of 90 days. The suspension was continued by subsequent orders dated June 28, Aug. 3, and Nov. 3, 1934, and was made indefinite by order of April 5, 1935. The last outstanding permit will expire by April 1938, except as it may have been qualified by discovery and application made for conversion into a lease.

Although a total of 322 potash-prospecting permits have been issued for land in New Mexico, only 160, involving 355,259 acres, were outstanding on Oct. 31, 1933, the date of the greatest number. On Jan. 31, 1936, there still remained 101, involving 217,448 acres. The initial two-year period on many of these have expired, but the permits are held active pending action on applications for extensions or affirmative action looking to their cancellation. Twelve Federal potash leases are now outstanding; nine in New Mexico and three in California.

For land in New Mexico several permits are in process of conversion to leases, with interest centered in an area southeast of Carlsbad containing deposits of langbeinite and sylvite. For Federal lands in the Salduro Desert of western Utah a few preference-right lease applications have been filed, but action on them has been suspended by request of the parties in interest, pending the formulation of an acceptable plan for unified development involving publicly owned and privately owned area.

In eastern Utah W. F. Reeder maintains his residence on certain asserted placer claims in Salt Valley, where, in 1924, he called attention to the formation of crystals in the casing of a well drilled for oil; these crystals were identified as carnallite and led to the finding of sylvite in the well cuttings. Sylvite was later found in cuttings from other wells in the vicinity, but as all were at considerable depth, no development work has resulted.

INCREASED USES

When potash imports were cut off in 1915 a concerted effort was made in industry to substitute sodium salts for potash salts wherever feasible. In some uses the substitutes have become firmly entrenched and have proved economical. Since American potash resources have been developed and the price greatly reduced, the benefits to be derived from a reversal of the wartime expedient warrant investigation concurrently with the search for new uses.

New and extended uses for potash are being found where improved quality is desired in the products of both agriculture and industry. With the restriction of agricultural production, more interest is shown in obtaining higher prices per acre by improving the quality of product, a factor too seldom recorded in fertilizer tests made throughout the country. Experiments to determine the effect of potash on the quality of agricultural products are now planned at the Kalisyndikat's agricultural experimental station in the Lechterfelde section of Berlin, where heretofore experiments on potash were confined largely to its effect in increasing the quantity. An increase in the use of potash is contemplated in the manufacture of glass fibers to increase its electrical resistance, and other ceramic products, where quality of product rather than price is the principal consideration⁷.

Increased use of potash is contemplated in the organization of the American Potash Institute, Inc., the advisability of which I mentioned two years ago, owing to the rapidly increasing capacity to produce potash in America. The formation of the Institute jointly by the producers and the principal importer of potash salts for the purpose of carrying on scientific and agricultural investigations to promote the efficient and profitable use of potash in crop production was announced on May 28, 1935. The plan of the American Potash Institute is to cooperate with State and Federal agencies and with agricultural organizations in the United States, Canada and Cuba. The directors consist of two representatives of each of the three principal producers in the United States and representatives of the principal importer. Each director has an equal voice in the affairs of the Institute. Offices were opened in Washington, D. C., on July 1, with a staff of about 20 people, which included a large number of the personnel and the excellent library of the scientific bureau of the N. V. Potash Export, My., Inc.

The Institute does not yet have any active interest in the use of potash in industry, which from the foregoing outline appears to present possibilities of additional demands for potash.

INTERNATIONAL SITUATION

The world potash supply in 1915, essentially under control of a single nation, was afforded for the fertilizer year ending April 30, 1935, by production in seven countries. In terms of pure potash equivalent the total output of 1,964,000 metric tons was divided as follows: 1,250,000 tons produced by six German companies, 350,000 tons by the French Government and one private company in Alsace, 140,000 tons by American producers, 100,000 tons by the three Spanish companies, 75,000 tons by the Soviet Union, 35,000 tons by the Polish Government, and 7000 tons in Palestine⁸. Of this tonnage, 1,735,000 tons was produced by concerns included in the international export pact headed by the Kalisyndikat. Two of the three Spanish producers did not enter into the Kalisyndikat agreement, or so-called "cartel," until they had established themselves as a factor in the world market through large sales at exceedingly low prices, which in America were reported as low as 27¢ a unit, as against previous prices of 60 to 70¢. This strategy resulted in a reported award to them of 15 per cent of the competitive market. With 4 per cent already awarded to Poland, Germany and France, sharing but 70 and 30 per cent, respectively, of the remaining market, now have 56.7 and 24.3 per cent of the total export markets. The Soviet Union and Palestine Potash Co., Inc. are reported to be working in cooperation with the cartel, leaving only the American output free of international trade understandings. The reduction in price stimulated sales, with the result

that the German output in 1934 attained record proportions, even though the dividends were materially reduced.

Statistics.—Potash produced and sold in the United States is reported in terms of merchantable salts, which range in purity from manure salts containing 25 per cent pure potash to potassium chloride of a purity suitable for the chemical trade, and includes various mixtures of these grades. An additional small tonnage of 20 per cent salts is recovered in the manufacture of cement. Information on imports for agriculture and industrial uses are published annually by the Bureau of Mines.

There being but two mines and two refineries in operation prior to 1935, there are no published statistics showing the output of our mines or refineries. Neither are there any statistics to differentiate between sales for agricultural and industrial uses of the domestic output. The production of potassium compounds made in America is published biannually by the Bureau of the Census, but these statistics for 1933 are limited to the output of potassium acetate, bitartrate, citrate hydroxide and iodide. Other compounds such as the nitrate, if any, and carbonate, are consolidated under a single item, as the Bureau does not reveal production of any compound when there are less than three producers.

Caution is necessary in comparing the statistical data on the value of potash fertilizer salts produced in America with those imported or exported, as basically all sales of potash salts of agricultural grade are at the same price quotations for the respective grades c.i.f. at Atlantic, Gulf, and Pacific ports to which price freight is added to interior delivery ports. Potash imported to America in 1934 was valued at an average of \$50.21 a ton of pure potash at seaboard. The domestic production for 1934 was valued at \$24.65, the realization at the points of production such as Carlsbad, N. Mex., and Trona, Calif., the difference being largely a matter of land and water freight charges to Atlantic seaboard ports from the mine or factory where it is produced. Export salts brought an apparent higher price than either import or domestic salts as the sales included the freight charges to the nearest port and in some instances may have included freight to foreign countries. The K_2O content of salts is not revealed but on the assumption that it averaged 50 per cent K_2O the pure potash exported was valued at \$65.62 a ton.

FOREIGN TRADE

Imports.—Ten years elapsed between the first production of mineral potash in Germany in 1861 and the shipment of 1400 tons of muriate to the United States in 1871. Tests of Stassfurt kainite were made at the Pennsylvania State College Agricultural Experiment Station in 1872⁹. This is the earliest definite reference to the use of kainite in America. Potash was used in earlier fertilizer experimental work at that station, but its source is not known.

The price paid for kainite in 1873 was $1\frac{1}{3}\text{¢}$ a pound and for muriate containing 85.30 per cent of K_2O , 3 to $3\frac{1}{2}\text{¢}$ a pound. In 1879 German exports of potash for agricultural purposes first exceeded those for industrial purposes. In Germany potash in agriculture began to exceed industrial uses in 1890.

Imports for agricultural purposes in the United States reached a maximum in 1930, with 322,370 tons of pure potash, valued at \$19,909,797, but dropped in 1934 to 154,735 tons, valued at \$7,769,286 and increased in 1935 to about 220,000 tons of K_2O valued at \$8,943,442.

Exports.—Much of the potash exported in the early days was derived from wood ashes as a byproduct of the clearing of land for habitation and was at times America's best source of foreign cash and credit. It permitted buying from foreign lands things that the colonists needed. This exported potash has been replaced through imports for agricultural purposes to the equivalent of 5,400,000 metric tons of pure potash during the last 30 years. Mention is often made of the dissipation of our soils first through the export of wood-ash potash and now of our farm crop, while seldom is an appraisal made of a greater loss of potash through soil leaching, soil erosion, fixation in the soils, and wastage into the ocean through our sewage systems.

In 1825 America exported potash from wood ashes valued at about \$2,000,000. It is again an important exporter of potash salts, but of mineral origin. According to the preliminary figures for 1935 it exported for agricultural purposes 75,985 short tons of potassium chloride and manure salts, valued at \$1,992,062, to 22 foreign countries¹⁰ as follows: Argentina, Belgium, Canada, China, Colombia, Cuba, British West Indies, Dominican Republic, Finland, Germany, Guatemala, Haiti, Honduras, Italy, Jamaica, Japan, Nicaragua, the Netherlands, New Zealand, Philippine Islands, Sweden and Venezuela. Of this amount, 50,176 short tons was shipped to Japan, 13,664 tons to Canada, and 8757 tons to Europe. Some of the material exported, such as that to Germany, was intended for transshipment to other countries, with the ultimate destination not revealed. Similarly, some of our exports consisted of imported potash salts that remained in American warehouses from the potash war of 1934–35.

The total exports of potash fertilizer salts in 1935 may be compared with 17,042 tons valued at \$643,367, in 1930, and 32,460 tons valued at \$1,267,109, in 1931, the previous record year for mineral potash.

Potash War of 1934–35.—The New Mexico manure salts became established as acceptable on the American market in 1931 and the muriate in 1932. The Spanish and Russian production became factors in 1933. In 1934 the war for world markets became a reality: the Spanish interests desired to establish large sales as a basis of future quotas; the Americans, to protect their investments; the cartel, to discourage competition and the

Soviet Union, to obtain international credit. The base price in 1934 was reduced from 60 to 40¢ a unit for muriate, with seasonal discounts as great as 12 per cent. This price was further cut by foreign producers in the spring of 1935, until potash was reported moving at as low as 27¢ a unit, and the price of manure salts became so low that American crude salts temporarily disappeared from the market, although the high-grade muriates became more firmly established. As a result of the Spanish producers joining the cartel and the cartel reaching a reported understanding with the Soviet producers, the American base price in July, 1935, of 45¢ a unit with seasonal discount is reported as being respected. The net result of the price war has been that the earnings of foreign producing companies were materially reduced and payment of dividends on the investments made in producing potash in America were delayed. The reduced price of potash did not reduce the quotations of mixed fertilizers, which increased with that of other commodities.

FOREIGN DEVELOPMENTS

Russia.—The fear in 1934 of a Russian invasion of American potash markets appears to have subsided with the reduction in price and with the understanding purported to have been reached by the Russians with the European cartel. The output at Solikamsk in 1933 of about 310,000 tons of salt was scheduled for an increase to 1,500,000 tons of crude salt for 1935. For the development of a potash deposit 32 miles south of Solikamsk, with double the capacity of the plant at Solikamsk, two development shafts are reported¹¹ to have been sunk and arrangements made for the sinking of a ventilating shaft.

Austria.—Potash is reported to have been discovered in prospecting the lower salt measures of the long-worked deposits in the Province of Salzburg, Austria, and thus another potential source of potash may be in the offing.

Ethiopia.—During the past year newspapers have often referred to potash as one of the important mineral resources of Ethiopia. It appears¹² that in 1911 a surface deposit of potassium chloride was observed about 60 miles south of Mersa Fatimari, or 10 miles south of the Eritrean frontier, on the Asoli salt plains, 127 m. below sea level. The potash occurs in brines brought to the surface by hot springs, which are estimated to be sufficient in volume to yield 5000 to 10,000 tons of potassium chloride annually. It is reported that each of five wells sunk to a depth of about 60 m. in a zone 12 km. long encountered steady flows of brine. To what extent the output may be increased by additional drilling is not yet known.

At the beginning of the World War this resource was considered the Allies' only assured source of potash. It was operated under a concession granted to Italian interests but production ceased in 1930 after a head

tax was assessed on the employees of the Italians. In transporting salts to the coast 1600 camels were used before a railroad was built through Eritrea, to within 7 miles of the deposit.

Poland.—Langbeinite, formerly mined in Germany and used in the manufacture of potassium sulfate in Poland, was introduced by Polish interests in 1933–34 as a fertilizer salt under the trade name “kalimag.” Its advantage, with 34 per cent of potassium sulfate and 53 per cent of magnesium sulfate, is its slow solubility. The popularity of kalimag is of interest to those prospecting in New Mexico, where a deposit of langbeinite ranging from 6 to 80 per cent pure, has been found.

Palestine.—The plant for the annual production of 10,000 tons of muriate at the north end of the Dead Sea in Palestine¹³ is being supplemented by a plant at the south end with a planned output of 100,000 tons.

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Salt Occurrences in the Potash Mines of New Mexico

BY RICHARD V. AGETON*

(New York Meeting, February, 1936)

SALT bodies in the form of rolls, horses (sometimes called horsebacks), folds, wants and pinches¹ have been encountered while driving entries and mining out rooms during the development of the potash mines in New Mexico. Superficially these sporadic salt occurrences appear to be much alike, but closer examination reveals individual characteristics, and after four years of observation it seems possible to divide them into three general groups that may tentatively be called "depositional," "erosional," and "structural." The data collected are not yet sufficient to permit forecasts of their probable location, so that the annoyance of having to mine salt instead of potash could be avoided or lessened. However, the information so far available appears to throw some light on the occurrence of the sylvite² bed now being mined, and it is as a progress report that the writer presents the results of his observations.

The mines of the United States Potash Co. and the Potash Company of America are about 20 miles northeast of Carlsbad, in Eddy County, N. M. They are on the southwestern flank of the Permian salt basin, which underlies western Texas, Oklahoma and Kansas, south-central Nebraska, southeastern Colorado and eastern New Mexico. The cuttings from many oil wells and the cores from the potash tests drilled in this basin indicate that this salt series approached its greatest thickness (about 4000 ft.) in the southeastern part of New Mexico and the adjoining portions of Texas, and that the most advanced stages of

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¹ The following definitions have been taken from A Glossary of the Mining and Mineral Industry, U. S. Bureau of Mines *Bulletin* 96:

Roll. An inequality in the roof or floor of a mine.

Horse (Horseback). A portion of the roof or floor which bulges or intrudes in the coal.

Fold. Rocks or strata that have been bent into domes and basins or rolls.

Want. A portion of coal seam in which the coal has been washed away and its place filled with clay or sand.

Pinch. A compression of the walls of a vein, or the roof and floor of a coal bed, which more or less completely displaces the ore or coal.

² A mixture of the minerals sylvite (KCl) and halite (NaCl).

desiccation were reached in marginal basins, of which four have already been recognized in Eddy County, New Mexico.

The evaporites containing these salt-bearing beds of Permian age have been divided into the upper or potash zone and the lower or non-potash zone. Lang³ has called the upper zone the "Salado halite," and the lower has been tentatively correlated by him with the Castile anhydrite. In the upper zone occur many beds of polyhalite and anhydrite and also beds of the more readily water-soluble salts of potash, such as sylvite, carnallite and langbeinite. None of the latter salts have yet been found in the lower zone.

The Permian salt group in the Carlsbad field has a regional dip of 2° to 3° SE. The deposits, although differing considerably locally, are in the main remarkably uniform in mineral composition. There is one easily recognizable sequence of white polyhalite, halite, and gray anhydrite that has been used in the correlation of the potentially commercial beds of potash salts. This sequence, from the top of the white polyhalite to the bottom of the gray anhydrite (25 to 40 ft.), has been recognized in all the cores and most of the oil-well cuttings in this region, and it can be traced throughout much of the western part of the Permian Basin. This key anhydrite bed, locally called "bed A," is from 45 to 80 ft. above the sylvite deposits that are being mined at the present time.

Contours, isopachs (lines representing equal thicknesses) and sections, using bed A as a guide, indicate that structural deformation resulted in a series of northeastward-plunging folds of relatively small size having their steepest dip to the northwest and long, gently dipping flanks to the southeast. Recent drilling about 8 miles east of the potash mines disclosed the presence of a dike or sill of igneous material about 40 ft. thick at a point in the upper salt series from 500 to 600 ft. below bed A. Such an intrusion might have had some effect in the deformation and alteration of the upper salts.

It is evident that the major part of the deformation causing these folds took place after the deposition of the potash-bearing salts. However, some folding is thought to have been contemporaneous with and some prior to the deposition. It is believed that the forming of these folds has had some effect on enrichment and impoverishment of the beds of more easily water-soluble potash salts, subsequent to their deposition. Core tests made in this part of the Permian salt basin have shown that in general, and particularly in the sylvite bed now being mined, there has been impoverishment within the bed both near the tops of the anticlines and near the bottoms of the synclines. In fact, cores taken through this bed, at what appear to be points near crests of anticlines and troughs of synclines, have contained none of the more easily water-soluble potash

³ W. B. Lang: Upper Permian Formation of Delaware Basin of Texas and New Mexico. *Amer. Assn. Petr. Geol. Bull.* (1935) **19**, 262-270.

minerals, even though the bed itself was well defined. At places, particularly on or near anticlinal folds, small fissures filled with salt and occasionally sylvite have been found in the anhydrite key bed A.

Marin⁴ mentions four factors which he believes to have been instrumental in the formation of the commercial potash deposits of Spain:

1. Movements within the beds themselves, caused by the weight of overlying sediments, so that the deposits move like fluids, owing to their plasticity.

2. Tectonic influences producing monoclinal or perhaps anticlinal folds without true synclines, thus leaving long sloping flanks away from the places of such tectonic disturbance and sharp folds nearer them. He gives as an example the absence of sylvite within the known zones on an anticline near Llobregat.

3. Ingress of comparatively fresh water into deposits, as a result of surface and subsurface erosion.

4. Readjustment of water of crystallization within the minerals themselves, and all readjustments of a chemical nature.

Conclusive evidence as to the mode of formation of the present commercial deposits in New Mexico is not yet available, but further investigation by competent geologists should throw more light on the subject.

NORMAL SEAM

A section along the rib of an entry driven through a normal portion of the sylvite bed is illustrated in Fig. 1. The irregular line at the bottom of the section shows the effects of mechanical undercutting. The bed being mined has certain regular characteristics. It is from 5 to 12 ft. thick and is composed mainly of sylvite. Negligible percentages of polyhalite, langbeinite, carnallite and other potash salts have also been found in the portions of this bed that have been mined. The face samples contain amounts of potash equivalent to 20 to 45 per cent K_2O . The sylvite crystals may be clear white, milky, bluish white, or reddish brown. Many of the milky white sylvite crystals are surrounded by the reddish brown sylvite, and in other places the entire crystal is reddish brown, giving the potash salts as mined a reddish brown color. This coloring is due to tiny particles of hematite.

Although both the roof and the floor of the bed are undulating, the waves or rolls are not high. The roof salts immediately above the bed have a clear apricot color, which is due to the inclusion of flakes and blebs of polyhalite having that color. The floor salts immediately below the bed are a deep reddish brown, because a larger amount of the reddish brown clay is included than is found in the sylvite bed itself.

⁴ A. Marin: *Historia de una molecula de potasa*. Inst. geol. y minero de Espana (1933).

At the base of the roof salt and also at the top of the floor salt there has been an accumulation of a greenish gray clay, next to the sylvite bed. These streaks of clayey salt are from 2 to 15 in. thick. Only a negligible quantity of this greenish gray clay is found in the sylvite bed itself, although in some places there are in the center of the bed streaks of this clayey salt from 2 to 6 in. thick. These clayey accumulations look like stylolites found in limestones and marbles, being irregular streaks along and transverse to the bedding.

The occurrence of these inclusions of greenish gray clay in the salt on both the roof and floor of this bed of sylvite accentuates the boundaries of the bed itself, and these inclusions were the main guides used in the study of the salt encountered in mining operations; in fact, similar accumulations of greenish gray clay have been noted by the writer in the salt both above and below all sylvite beds of possible commercial importance in the potash area.

These clayey salts vary in color from gray to greenish gray, in composition from salty clay (60 to 80 per cent clay and 40 to 20 per cent salt) to clayey salt (10 to 40 per cent clay and 60 to 90 per cent salt), and in thickness from $\frac{1}{2}$ in. to 10 in. In general, the thicker the clay accumulation, the lower the clay content.

K. Jacobi and J. P. Smith⁵, of the United States Potash Co., after a chemical and mineralogic examination of samples of these clay accumulations, reached the conclusion that: (1) little if any humus is present, (2) the clay has not been exposed to much oxidation, (3) it is probably a wind-blown silt that settled in water or solution of moderate depth, (4) it has characteristics that indicate a combination of eolian and water-deposited origin, and (5) the particular samples submitted contain 4.5 to 5.0 per cent of greenish gray clay and the balance halite, with a very few fragmental particles of polyhalite, sylvite, langbeinite and carnallite.

Schaller and Henderson⁶, describing the clays found in the Permian salts, note the occurrence of three types of clay in the potash-bearing beds: (1) a reddish brown clay, associated with the potash; (2) a dull greenish clay; and (3) a gray to white clay (containing magnesite), generally associated with the anhydrite. The reddish brown clay contains about 5.4 parts of the ferric oxide (Fe_2O_3) to 1 part of the ferrous oxide (FeO). The greenish gray clay contains only 0.3 part of the ferric oxide to 1 part of the ferrous oxide. Schaller and Henderson (*idem*, 37, 38) observed that the hematite (ferric oxide) is generally restricted to the potash-bearing minerals and suggested that it might be due to bacterial growths, though they did not study the relation of such growths to the hematite.

⁵ Personal communication, April 24, 1935.

⁶ W. T. Schaller and E. P. Henderson: Mineralogy of Drill Cores from the Potash Field of New Mexico and Texas. U.S. Geol. Survey *Bull.* 833 (1932) 24.

Marin, in describing the natural conditions requisite to the formation of the potash deposits of Spain, cites the following:

1. Formation of intermittent lagoonal regions as the result of alternating periods of uplift and depression, tending to form shallow bars that would alternately cut off and allow ingress of fresh supplies of ocean water.

2. An extremely arid climate for a considerable period, which would give rise to rapid desiccation.

3. Intermittent flows of comparatively fresh water containing silt, sulfates, chlorides and carbonates into such occasionally landlocked estuaries to provide: (1) an enrichment of the residual ocean water from an intensive sporadic erosion of the surrounding terrane, such as would occur during intensive dehydration; (2) an impermeable layer of mud or silt (clay and shale) over the beds of soluble potash salts, to prevent their subsequent erosion upon a change in climatic conditions.

Deposition under such circumstances, where the potash seams with their underlying saline beds were laid down upon an undulating bottom that sloped toward a sea or lagoon, might result, on a further tilting of the beds already deposited, in the formation of minor folds. Some folding might also have been caused by loading. As sylvite is not deposited directly from desiccation of ocean water, vertical loads on the crests of anticlines or the troughs of synclines might tend to squeeze the solutions with the potash salts in the spongelike salt mass toward the adjacent limbs of the folds and thus alter the deposits in such places. The deposition of overlying beds along the margin of a comparatively small lake or lagoon might also cause the formation of minor folds in the center of the basin. Something similar to one or both of these actions appears to have caused the minor structural salt occurrences noted near present major synclines. However, some of the salt occurrences noted were both depositional and erosional. Stream beds, islands and bars are continually being formed in the salt lakes of Utah, Nevada and California at the present time. A subsequent deposition of barren salt over the top of a deposit containing potash would result in the formation of salt horses in such stream beds and around such islands and bars in the potash bed. The deposition of potash-bearing salts in such stream beds and around islands and bars in a salt bed would result in the formation of potash horses within the barren salts.

DEPOSITIONAL HORSES

A sketch of a salt horse apparently of depositional type is shown in Fig. 2. This is here referred to the bar type because of its resemblance to the bars so often piled up on the beaches of saline lakes, particularly after a winter storm. The deposit slopes gradually on one side, as if toward the source of the salt, whereas on the other (shoreward) side it

breaks off abruptly. The material in the bar itself is in general very different from either the roof or the floor salt, and it contains large cakes of clear white salt, some of them 12 in. thick, 3 ft. wide and 4 to 6 ft. long. The clear white salt contains very few, if any, impurities. The

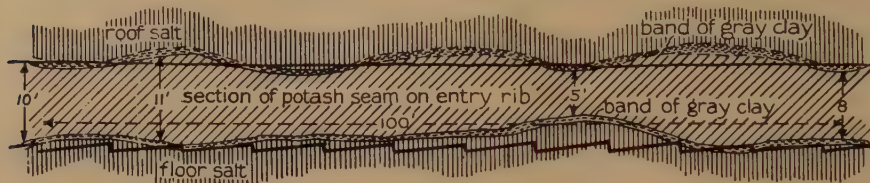


FIG. 1.—NORMAL SEAM.

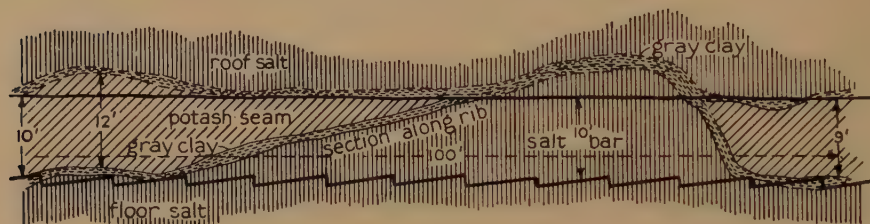


FIG. 2.—DEPOSITIONAL SALT HORSE, BAR TYPE.

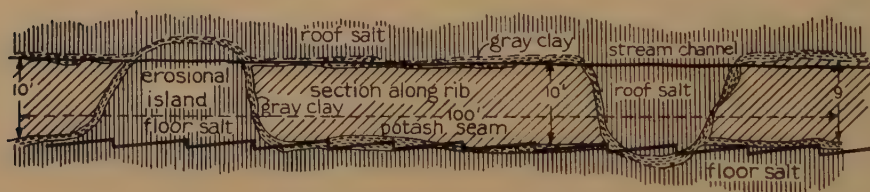


FIG. 3.—EROSIONAL SALT HORSES.

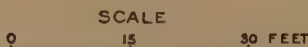
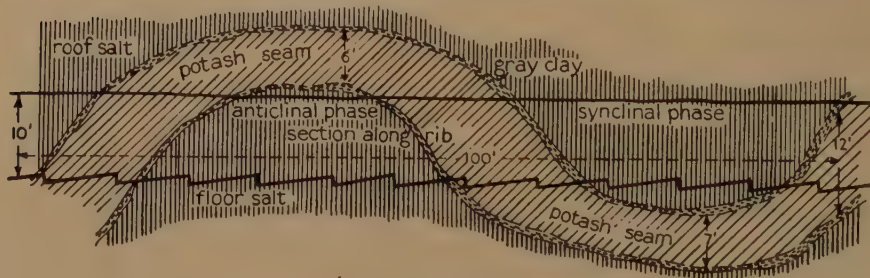


FIG. 4.—STRUCTURAL SALT HORSES.

bar is apparently a shore-line deposit at a particular stage of a lake, or sea, and differs greatly from the other salt horses found. This bar is about 85 ft. wide and 124 ft. long and had a maximum estimated height of 12 ft. The contact between the bar salt and the sylvite bed was

plain. On all sides exposed there was an accumulation of the greenish gray clayey halite (from 2 to 6 in. thick) between the material composing the bar and the sylvite bed. Furthermore, although the sylvite bed contained the equivalent of 25 to 30 per cent K_2O right up to this clayey halite, only a few blebs of sylvite were found in either the bar material or the clayey halite.

EROSIONAL HORSES

The so-called "erosional horses" or "wants" can be divided into two types—erosional islands and stream channels. Sketches of these are shown in Fig. 3.

The "erosional island" resembles the bar type in that it extends upward into and in places through the sylvite. However, careful examination of most of the salt horses of the island type shows that they are of the same composition and structure as the floor salt. In some places, however, they have the same type of structure (large salt cakes) as the deposits of the bar type, suggesting that some of these salt occurrences were probably formed by the partial erosion of a bar or reef of salt prior to the deposition of the sylvite bed. In nearly every occurrence the sides of the island approach the vertical. The seamlike inclusions of greenish gray clayey salt are also found at the contact between the salt horse and the sylvite bed. Usually the material of the island contains little if any sylvite. These islands may measure as much as 40 ft. in diameter, but generally they are smaller.

The stream channels (Fig. 3) appear on first inspection very similar to the islands. However, closer examination shows that the material in the horse or want is similar to that in the roof salt and does not contain the large blocks of clear white salt found in the bar deposits. Both the roof salt and the channel salts in all probability were deposited at the same time. The greenish gray clayey salt continues beneath the bottom of the horse, separating the salt in the horse from the sylvite bed. Like the other types of salt horses, the channel deposits contain but little sylvite. In general these stream channels are comparatively narrow, but their length has not been determined.

STRUCTURAL SALT HORSES

The structural type of salt horse is by far the most interesting. It is a minor structural development or fold and so far has been found only near major synclines. This type is unlike the other types described in that the sylvite bed is continuous; it merely disappears either above or below the working level. In the anticlinal phase (Fig. 4) the whole bed, inclined at a steep angle, extends upward beyond the face of the working place but can be found in its entirety in the roof. In the synclinal phase

the bed dips down at an angle with the face of the room or entry and disappears into the floor. In both phases the accumulations of greenish gray, clayey salt on the roof and floor of the sylvite bed are present and, though the bed itself is slightly thinner, both on the top of the anticlinal phase and on the bottom of the synclinal phase, it is not cut out, as in the salt occurrences of the depositional or erosional types. In one locality a drop in the roof of the bed of 17 ft. in 60 ft. along the entry was noted.

Brief examinations indicate a flowage within the sylvite bed itself. Individual crystals of sylvite are larger and longer than those found in portions of the bed that have not been subjected to so much deformation. Some large crystals of pure white sylvite have been found both in the roof salt above an anticlinal phase and in the floor salt below a synclinal phase.

Deposition of overlying beds along the margin of a comparatively small lake or lagoon might cause the formation of such minor folds in the center of a basin.

There are still other types of salt occurrences that may have been caused either by depositional or structural action. In one of these the sylvite in the bed feathers completely out, leaving nothing but salt. This condition may continue for more than 100 ft., and in these portions the bed appears normal except that it contains only small amounts of polyhalite, sylvite and greenish gray clay. These salt occurrences have been found in what appears to be the bottom of the major synclinal phase of one of the northeastward-plunging folds mentioned above. It may be that continued deposition of overlying beds upon the center of a small lagoonal basin would create sufficient pressure to cause a movement of the highly impregnated solutions contained in the spongelike salt mass away from the zone of pressure toward the edges of the lake or lagoon.

Subsequent structural deformations sufficient to cause the formation of the northeastward-plunging folds might have the same action at the tops of anticlines and the bottoms of synclines.

Recent tests made in the United States Bureau of Mines laboratories to determine the crushing strength of the material in the sylvite bed developed the fact that an appreciable quantity of water is still present in the sylvite. This water contains salts of sodium, potassium, magnesium and calcium. The migration or movement of such a saturated solution through the crystalline spongelike mass of the potash salts as laid down could have been caused in several ways—for example, it may have been due to the continued action of structural deformation. Any deformation, structural or depositional, might facilitate the migration of sylvite from the places of greatest load (tops of anticlines and bottoms of synclines) to intermediate locations.

That enrichment has played an important part in the formation of the commercial beds of sylvinite now being mined is indicated from the reports of Schaller and Henderson⁷ on their studies of the minerals found in the cores. They state:

Many of the mineral associations now present in the deposit, as represented by pieces of core, . . . suggest that large proportions of these minerals do not represent the primary minerals originally laid down or precipitated in the Permian waters but consist of reaction products formed by the later action of different kinds of solutions on the first minerals that formed.

They cite as examples the alteration of anhydrite to polyhalite and of sylvite to langbeinite and state further that moving water must have been the universal medium in which these changes took place.

Just what influence desiccation, further influx of sea water, torrential rains, inflowing streams of comparatively fresh water conveying sulfates, carbonates and chlorides, bacterial and algal growths, climatic and seasonal changes, intrusion of igneous material into the salt bed, and structural changes caused by such intrusions had on the formation of the commercial sylvite beds of the Permian deposits can be determined only by further and much more detailed study of data and material now at hand or to be gathered as prospecting, development and production continue.

DISCUSSION

(*E. F. Burchard presiding*)

G. R. MANSFIELD,* Washington, D. C.—Mr. Ageton in the course of his duties has had frequent access to the two potash mines, whereas I have had opportunity to visit them only once. Therefore, I am somewhat hesitant in venturing to differ with his interpretation of the observed facts. His interpretation seems to assume that the potash salts were produced by desiccation at the surface and were then eroded or acted upon by surface agencies. However, the study of the minerals in the potash cores by Schaller and Henderson shows so many changes by replacement by circulating solutions at greater or less depth that I doubt if the potash salts could have been exposed at the surface at any time. In the mines I was struck by the resemblance that the clay bands which, though apparently regular, are highly irregular in detail, bear to the stylolites that are so commonly seen in marbles and that are attributed to settling or slumping clay beds or zones where solution has taken place. It seems to me more likely that the clay bands have accumulated by the solution of the salt in which it was at first disseminated, and that as solution in favorable zones continued there was a settling and rearrangement of the salt and potash beds sufficient to produce the effects of erosion and accumulation described in the paper.

E. S. MOORE,† Toronto, Canada.—It occurs to me, in looking at the sections, that if there is any evidence of slight compression in the area there might be an occurrence

⁷ W. T. Schaller and E. P. Henderson: Reference of footnote 6, pages 13, 14.

* Geologist, U. S. Geological Survey.

† Professor of Economic Geology, University of Toronto.

similar to that in many coal mines, where veins and pockets of clay are squeezed up into the coal from underlying clay beds.

E. F. BURCHARD,* Washington, D. C.—Can you tell us, Mr. Smith, whether there is such a condition as Mr. Moore has suggested?

H. I. SMITH,† Washington, D. C.—It is very hard to say just what the condition is. I do not think it is the same as in coal mines, but I think there may have been some dissolving of the salt and later the salt from above or below may have squeezed into the holes.

E. F. BURCHARD.—With such folds it seems as though some of the salt might have been squeezed up into the smaller crevices.

W. T. SCHALLER,‡ Washington, D. C.—I have been in this mine, but not since the features shown by Mr. Ageton were well developed. I am inclined to disagree with Mr. Ageton in his interpretation and to agree with Mr. Mansfield. The potash seams consist of about one-half potassium chloride and one-half sodium chloride, and potassium chloride does not separate or form by the evaporation of sea water; there is too much magnesium chloride in sea water to permit the potassium chloride to come out first. Therefore, all the potassium chloride in the mine is the result of secondary processes, and I am inclined to believe that the structural features shown can be explained otherwise or that Mr. Ageton did not give enough attention to the possibility that they may result from recrystallization and rearrangement of the deposit as a whole after its formation. And I am inclined to disagree with the interpretation that these are bars and islands and channels.

MEMBER.—Is there any evidence or earmark of the salt following the line of the contour of potash?

G. R. MANSFIELD.—In regard to the structural features, I think there has been a certain amount of change in the original structure, not only in a large way but also in the development of minor folds. As far as we know, there are several of these undulations, as I suppose they might be called, on the edge of the large salt-bearing basin. Now there have been certain opportunities, for these beds have been slightly folded, for parts of the beds to become squeezed a little, and for other parts to be expanded, and since these salts are more or less mobile there might be some attendant thickening up in one place and a thinning out in another. In places where we should expect the thickening to occur—that is, on the tops of the arches and the bottoms of the troughs—the quality of the potash seems to have become inferior; that is, the potash diminishes in quantity and goes out, whereas on the sides it seems to be richer. One would suppose that, in certain types of deposits at least, the opposite might be true, that the accumulation would be thicker and richer in the tops and the bottoms of the folds. The reason for the condition noted here may be that where the folds open, on the top and in the trough, perhaps there is a greater opportunity for the passage of the solutions and for replacements and for the solution to remove the potash. Another element in the position is probably the compaction produced by the settlement and readjustment of the salt beds, as already suggested.

MEMBER.—Are the bands of salt disrupted or are they continuous?

* Mining Geologist, U. S. Geological Survey.

† Chief, Mineral Leasing Division, U. S. Geological Survey.

‡ Mineralogical Chemist, U. S. Geological Survey.

G. R. MANSFIELD.—It is not possible to see much above or below the actual potash bed that is being operated, but in this general area we see sodium chloride and clay material, and then the mixture of sodium and potassium chloride forming a sylvite bed. In general, its color is apricot or reddish, approaching a brick red color, and it is more or less blotchy with white or slightly bluish tinted sylvite, surrounded by rather strong red irregular masses of the same mineral, and they in turn become intermeshed with the sodium chloride.

The potassium content in terms of K_2O runs around 28 to 29 per cent, but if a section were made through the sylvite bed there would not be a band of this and a band of that; there would be a sort of mixture of reds and whites and apricot coloring, but the whole would give an approximate average potash content such as that stated.

MEMBERS.—Some of the geologists working on the domes in Europe and the East have brought out information about differential flow of salt and potash units. Potash seemed to flow more freely than salt and so form enrichments, as it were, so that it was very hard or very difficult to predict the structure.

C. H. BEHRE, JR., * Evanston, Ill.—I should like to ask whether there is any evidence of secondary salts in connection with this unusual occurrence of the sylvite. In German publications there are frequently shown structural features of much the same type as those that have been interpreted as secondary as a result of the difference in solubility.

W. T. SCHALLER.—It cannot be primary therefore it must be secondary, so I think your question is answered.

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Applications of Potash in the Ceramic Industry

BY NELSON W. TAYLOR*

(New York Meeting, February, 1936)

WITH the extensive deposits of potash minerals which have been discovered in the southwestern states, and their rapid development, a permanent American supply of potassium compounds is now assured. The 325-year history of the potash industry in this country has been told by H. I. Smith¹, of the U.S. Geological Survey, and the more recent activities of the Survey and of American potash companies were described last year by the same author². In fact, it may be stated that so great is the productive capacity that two mines in New Mexico and one in California are able to supply more than half a million tons of crude and refined salts over and above that produced for agricultural purposes. The era of scarcity has given way to an era of superabundance.

This situation makes it desirable for the potash producer to look for other applications for his product, and it is with this in mind that the writer presents some information pertaining to present and possible future uses for potassium compounds in the ceramic industry.

CERAMIC PRODUCTS

Ceramic products are essentially solid materials containing one or more oxides such as SiO_2 , Al_2O_3 , CaO , MgO , FeO , K_2O , Na_2O , TiO_2 , which are the oxides of greatest abundance in the earth's crust. The typical ceramic process is that of firing or burning the batch, and the product may be glassy, crystalline or mixed. In the glassy or vitreous group come glass, porcelain enamel and glazes; in the crystalline group, refractory materials such as silica brick, or chromite or magnesia; and in the mixed group, porcelain, chinaware, stoneware, tile, brick, etc., which contain both a vitreous and a crystalline phase, in variable proportions. The ceramic industry of the United States is a billion dollar industry, and is important not only because of its magnitude but also because of the key position occupied by certain ceramic products in the whole manufacturing structure.

From the standpoint of use of potash, the field of refractories is of no great interest, since K_2O lowers the melting point of a refractory oxide and

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¹ H. I. Smith: *Eng. & Min. Jnl.* (1933) 134.

² H. I. Smith: This volume, page 329.

thereby lowers its value. In mixed products such as sanitary and electrical porcelain, chinaware, stoneware, tile and vitrified brick, the strength lies principally in the glassy phase. The viscosity of this glass must be very high when vitrification occurs, so that the shape of the piece is not deformed. Potash feldspar has an extremely high viscosity, even at temperatures well above its melting point, and for this reason this feldspar finds extensive use, and is regarded as one of the principal raw materials of the industry. The viscosity does not depend so much on K_2O alone as on the proportion of $K_2O:Al_2O_3:SiO_2$, but one can only say that at a given temperature the potash feldspar orthoclase will have a distinctly higher viscosity than the corresponding soda compound³. In certain electrical applications, such as electrical porcelain, the lower electrical conductivity of the porcelain containing potassium, as compared with the sodium porcelain, is also a distinct advantage. It should be borne in mind, however, that the principal source of potassium in these products at present is feldspar.

VITREOUS CERAMIC PRODUCTS

Under this heading are to be included glasses, enamels and glazes. This group of products furnishes the most likely market for potassium compounds, since a certain amount of alkali flux is needed in their composition.

Glass

The alkali content of commercial glasses depends on the type. For window glass, about 15 per cent soda is normal. Glass for containers requires 15 to 17 per cent soda. Thin blown glass, as in bulbs, runs somewhat higher in soda, approximately 20 per cent. This probably represents the maximum alkali in any commercial glass. The approximate alkali content is determined largely by the composition of the eutectic in the $Na_2O-CaO-SiO_2$ system, although commercial glasses always contain significant amounts of other oxides.

In several ways the presence of potash confers desirable properties on glass but its greater cost has necessarily limited its extensive application. In certain types of glass products that have high intrinsic value because of shape or other properties, the cost of the raw materials is not an important factor, and in such cases we may see how potash is used, and what advantages its use confers. With a fall in price a more extensive application of potash to cheaper glass products in higher tonnages will certainly follow.

At present, the applications of potash to glass are primarily in the form of potassium carbonate. As explained later, the potash-lead glasses require the nitrate.

³ N. L. Bowen: Viscosity Data for Silicate Melts. Geophysical Laboratory *Pub.* 849.

Tableware.—The best quality of crystal tableware is generally a straight potash-lead-silica glass. The glass has a high brilliance and a good musical tone or ring when struck. The potash content ranges up to 7 or 8 per cent. The tone, or sonority, is apparently due to potash, for the corresponding soda glasses are less satisfactory from this standpoint. In addition to these properties of brilliance and tone, a good chemical durability or resistance to weathering is to be ascribed to the presence of potash rather than soda. Because of the easy reduction of lead oxide in the founding process, potassium nitrate is almost universally used to maintain oxidizing conditions. The use of potassium carbonate is less important.

X-ray Lead Glass and Other Potash-lead Glasses.—Because of the high absorption of X-rays by lead, glasses of high lead content are being made. The alkali is practically all potash, because of high brilliance and weather resistance. The brilliance of high-lead glasses containing potash is a desirable feature for architectural applications, church windows, sculptured glass, etc. A drop in price of lead and potash undoubtedly would stimulate developments in this field.

Optical Flint Glass.—In optical glasses the potash content is customarily about twice as much as the soda content. This gives good durability and color. The ratio, 2 to 1, is slightly less than the optimum, which is about 7 to 3. The average potash content of several flint glasses is about 7 per cent. In one or two kinds of crown glass the potash content rises to 16 per cent.

The worst offender in regard to color purity is ferrous iron, which gives a green tinge at concentrations even as low as 0.01 per cent, but the effect of iron is much less harmful in a potash-soda glass than in the corresponding high-soda optical glass.

Ultraviolet Transmitting Glass.—As the result of observations on samples of relatively iron-free glass prepared by the U.S. Bureau of Standards, it has been found that soda-lime-silica and soda-silica glasses are unstable photochemically, while potash-lime-silica glass is not appreciably affected by ultraviolet radiation. It, therefore, appears that soda is the photosensitive constituent in a soda-lime-silica glass. The recent developments in the field of high intensity illumination may call for more extensive use of potash glass to inhibit solarization effects.

Electrical Insulators.—It is difficult to find any relation between composition and electrical properties of glass for all of the conditions of voltage, frequency, etc., to which the glass might be subjected. The scientific literature does not furnish sufficient data to permit one to draw final conclusions. However, certain investigations by Gehlhoff and Thomas⁴ on the variation of electric conductivity in the temperature

⁴ G. Gehlhoff and M. Thomas: *Ztsch. tech. Physik* (1925) 6, 544-54.

range 100° to 400° C., for a number of glasses of the general composition Na_2O (or K_2O). MeO . 6SiO_2 , in which MeO represents MgO , CaO , ZnO , BaO and PbO , show an insulating advantage of K_2O over Na_2O . Glasses containing both Na_2O and K_2O gave poorer conduction than either alone, the minimum conductivity being at a $\text{Na}_2\text{O}:\text{K}_2\text{O}$ ratio of about 1:4. There is a great need for further study, to ascertain the specific effects of the various components, the part played by each in the process of conduction, and the effect of temperature on the conduction process. This problem is becoming increasingly important in view of the new developments in spun glass (very fine glass fibers) which show great promise as protective insulating material instead of cotton on copper wires. This application may represent a considerable tonnage of glass. Chemical durability also becomes an important factor in spun glass. The alkali must be chosen to give the best resistance to moisture attack. A proper balance between soda and potash is necessary.

Colored Glass.—R. S. Scholes⁵ presents certain interesting information pertaining to the influence of potash as compared with soda on colored oxides in glass:

Manganese in a potash glass produces a blue violet. In soda glass, manganese produces a red violet, tending toward brown in high concentrations. Similarly, nickel oxide gives a deep blue-violet in the potash glasses, but a rather dull, brownish violet in the soda glasses. These facts are important in connection with decolorizing. In the potash glasses, much less cobalt oxide is required in connection with manganese to secure a good neutral tint, or balance of color, for crystal glass. Likewise nickel is a suitable decolorizer for glasses high in potash, whereas its effect in the soda glasses is decidedly ugly. In the potash glasses selenium produces a true rose color, or cerise, which may have violet tints, but in the soda glasses, similar concentrations of selenium yield only a salmon pink, tending toward yellow or amber rather than toward violet.

Double Glazing.—For purposes of insulation, two sheets of glass are placed in parallel position, separated by a suitable spacer. This type of construction has excellent resistance to heat transfer and is designed for use in large refrigerators and in air-conditioning of buildings. When ordinary polished plate glass or window glass sheets are used in structures of this kind, a white alkaline dust forms in the course of time on the interior surfaces of the glass plates, unless provision is made for the free circulation of air between the sheets. The dust or efflorescence clouds the sheets so that their transparency is impaired. The circulation of air between the sheets involves complications which reduce the efficiency of the insulation. On the other hand, by a modification of the glass-batch composition, the dusting or scumming difficulty may be eliminated without sacrifice of strength or hardness of the glass. Satisfactory composi-

⁵ R. S. Scholes: Uses of Potassium Carbonate in Glass Making. *Ceramic Industry* (1935) 24, 83.

tions require the use of about 6.5 per cent K_2O , but the content may run up to 10 per cent by weight. It appears that the surfaces of glass contain approximately 0.06 gram water per square foot of surface, irrespective of whether these glasses be soda-lime-silica or soda-lime-potassium-silica. The presence of the potash inhibits the formation of crystals, perhaps because the potassium crystals are more soluble or more hygroscopic than the soda crystals. The mechanism is as yet uncertain, but the fact remains that potash-bearing glass does confer distinctly advantageous properties for double glazing. This new development of the sheet-glass industry may in time reach significant proportions because it is a step in the right direction toward better control of temperature and humidity in public and private buildings⁶.

Chemically Durable Glasses.—The problem of chemical durability in the glass industry is analogous to that of corrosion resistance in the metal industry. The term "chemical durability" is an inverse measure of the rate of surface destruction by water, acids or other common weathering agents. It is generally recognized that, other things being equal, a potash-soda glass is superior in its durability to the corresponding soda glass. There are, however, other methods of protection involving change of composition, such as an increase in the alumina content or in the boric oxide content at the expense of the alkali. These methods are usually cheaper than that involving potash, and unless the price of potash is considerably reduced its extensive application for this particular purpose is not expected.

VITREOUS ENAMELS

The past 10 years have been marked by great expansion in the porcelain enamel field, where a vitreous enamel composition is melted on to the surface of sheet metal. This enamel is really a glass of melting temperature lower than ordinary window sheet or bottle glass. It is used as a protective and decorative surface on refrigerators, stoves, washtubs, electric signs, store fronts, etc. Broadly speaking, there are four enamel types:

1. The so-called sheet-steel ground coats, which contain about 7 per cent soda ash, added as such, and about 4 per cent niter ($NaNO_3$).
2. The cover coats, containing roughly the same amount of these constituents. Taking the feldspar into account the total percentage ($Na_2O + K_2O$) is close to 20 per cent.
3. Enamels for cast iron, which may be divided into wet-process cast-iron enamels and dry-process cast-iron enamels. The former require about 4 per cent soda ash added as such; in the latter the soda ash content is almost negligible. In dry-process cover-coat enamels the soda ash

⁶ J. C. Parkinson: U.S. Patent 1954058 (April 10, 1934).

content is about 4 or 5 per cent. The combined content of soda and potash in these enamels is about 15 per cent. The only source of potash, however, is the feldspar.

4. Enamels for jewelry, where high brilliance and luster are desired. They are essentially potassium lead silicates and call for as much as 36 per cent K_2O in the product or 54 per cent KNO_3 in the batch formula. This group does not represent an important tonnage.

In general, it may be said that very little potash as such is used in vitreous enamels because soda is so much cheaper. However, the two may be interchanged in equivalent amounts without significant change in properties. There is some indication that potash increases the brilliance or luster of enamels, which is a very desirable feature in enamels for steel and cast iron. If this is so, it would increase its use, if the price is low enough to justify it. The potash probably will have to compete with barium, however, as this element is also very effective in this regard.

GLAZES

Glazes are vitreous coatings applied to ceramic bodies such as china-ware, porcelain, etc. In composition, they are alkali-alumina-lead silicates or alkali-alumina borosilicates. Their purpose is to give decoration and protection, and it is necessary that they possess physical properties of elasticity, strength and expansion to enable them to fit the body. While potash is an ingredient in many glaze compositions, it is derived usually from feldspar and is not expected to represent an important application of other potassium compounds.

On the other hand, an effort is being made to replace the lead oxide in glazes by some nonpoisonous fluxing material. One of the properties contributed by lead oxide is brilliance or luster. Potash has some value along this line, and it may prove to have a useful application in this direction.

The practice of salt glazing by throwing sodium chloride into the hot kiln so that the vapors will attack the hot ware and produce complex silicates, is widely used. Excellent glazes have been made by dipping the ware into sodium chloride solution or into a mixture of sodium chloride and other glaze components, before firing. It is possible that potassium chloride may give superior results under similar conditions.

NECESSITY FOR RESEARCH

It will be clear from this presentation that the ceramic industry furnishes an important field of application of potassium compounds (principally carbonate and nitrate). Several of the leads are sufficiently interesting to justify further study. The opportunity is waiting for a live organization to organize a program of experimental research.

DISCUSSION

(H. S. Spence presiding)

H. I. SMITH,* Washington, D. C.—Why is it that sodium carbonate is listed as \$22 and potassium at \$140 a ton?

N. W. TAYLOR.—Sodium salts are almost universally distributed in nature and the production of sodium salt by processes such as the Solvay is a far more economical procedure than the production of potash. The point I wanted to make in general about the ceramic applications is that hitherto sodium salts have been very much cheaper than potassium salts, and so the present state of the industry as regards these salts reflects that difference. The differential if reduced would surely bring about new applications. In view of the largely increased and increasing production of potash in the United States, should we not be due for a considerable reduction in potash prices?

P. M. TYLER,† Washington, D. C.—I should like to hear a little more about the competition between potash in the form of a potash salt and potash as contained in feldspar, and to make it a little harder, I would like to ask Mr. Taylor if he has given thought to the addition of still another base in the form of lithia from the use of lepidolite.

N. W. TAYLOR.—Of course, the reason that feldspar is used so extensively in the ceramic industries is because it does not require any modification. It can be used directly. There is a certain amount of purification required for the elimination of certain accessory minerals, but otherwise it is ready for use. Feldspar has a high viscosity intrinsically, and that is an essential property of ceramic materials. Such materials must be brought to a state of fusion without change in shape of the object occurring. That is the special advantage of feldspar.

I tried to point out that the potash K_2O intrinsically has certain advantageous properties. In those cases it seemed to me that potash from New Mexico or related sources would have real application, but I do not anticipate any likelihood of making a synthetic feldspar, starting with potash from New Mexico as a source.

Regarding the lepidolite question, sometimes lepidolite can be used in the glass industry, and lepidolite would be used more extensively, I think, if cheaper sources were available.

P. M. TYLER.—Do I understand that you think it would be more economical to use feldspar whenever possible—whenever the application will carry the accompanying alumina and silica?

N. W. TAYLOR.—I think so. There may be cases, of course, where the ratio of potash to silica in the final product does not conform to the feldspar ratio, but is higher.

P. M. TYLER.—I was a little surprised at your recommendations in view of the fact that there is a tendency to use a natural compound that has already been formed rather than to develop a synthetic compound in the batch.

N. W. TAYLOR.—Feldspar has been very satisfactory in recent years in satisfying the alumina and alkali requirements. It would appear to be an economical form in which to buy alumina and alkali. Nepheline syenite is also showing promise.

* Chief, Mineral Leasing Division, U. S. Geol. Survey.

† Chief Engineer, Metals and Nonmetals Division, U. S. Bureau of Mines.

MEMBER.—In what type of glass is lithia used?

N. W. TAYLOR.—Chiefly certain varieties of the Pyrex type, for transmitting ultraviolet rays.

W. T. SCHALLER,* Washington, D. C.—A few years ago the main object of adding lithia was to make the opal tinted glass used for table tops, and it had a considerable development until it was used to face the front of buildings and the sunlight turned it pink irregularly.

P. M. TYLER.—It is my understanding that the vogue for lithia has declined, and I attribute it not only to the action to which Mr. Schaller referred but to the development of other more effective opacifying materials. There is an effort being made to expand again the use of lepidolite, and more particularly a new product, relatively new to this market, that comes from South Africa, which contains 4.25 or 4.5 per cent lithia instead of the 2 per cent that is characteristic of our North American material.

* Mineralogical Chemist, U. S. Geological Survey.

Gravimetric Survey of the Malagash Salt Deposit, Nova Scotia

BY A. H. MILLER* AND G. W. H. NORMAN†

(New York Meeting, February, 1936)

THIS survey is one of the more recent tests of geophysical methods of prospecting by the Dominion Observatory and the Geological Survey of Canada, of which the purpose is to find out what application such methods may have to geological problems in Canada. The Malagash salt deposit lies on the north shore of Nova Scotia about 40 miles east of the New Brunswick boundary, and forms a part of faulted and folded Carboniferous strata. The salt beds are exposed only in the mine workings, which are about 1000 ft. deep and at one place extend for 1000 ft. along the strike. Lack of rock exposures, complicated folds, and faults make it impossible to predict the location of salt away from the mine. A survey of an area in the vicinity of the mine was therefore undertaken to find out whether the salt could be located by the torsion balance and whether its extension from the mine workings could be traced. As the gravitational effects at Malagash may be fairly typical of those at other concealed salt deposits in the Carboniferous rocks of the Maritime Provinces of Canada, the survey has a direct bearing on prospecting for other deposits.

The survey was made in 1934 by A. H. Miller in collaboration with Dr. W. A. Bell, of the Geological Survey of Canada. Observations began on June 15 and were completed four months later, on Oct. 15. Definite gravitational effects including a lowering of gravity were observed over the deposit. The survey also disclosed one other area of low gravity east of the mine but whether this area is underlain by a large body of salt is not yet established.

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GEOLOGY OF THE DEPOSIT¹

The consolidated rocks at Malagash are a part of the Cumberland Basin of folded Carboniferous strata. This basin is geosynclinal in structure and extends eastward from Chignecto Bay across the northern part of Nova Scotia. The Carboniferous strata of the basin consist of a lower group, Mississippian in age, of soft, prevailing red shales and some sandstone with intercalated beds of gypsum, anhydrite, salt, and fossiliferous marine limestone, and an upper group, Pennsylvanian in age, of sandstone, conglomerate and shale, in places containing coal seams. The Mississippian strata are brought to the surface only along anticlinal folds, and are poorly exposed on account of their chemical and physical character. The anticlinal structure that brings the Mississippian rocks to the surface at Malagash extends southwestward from Malagash for about 40 miles. The south limb of this anticline dips steeply south at 45° to 75°. The north limb is overturned in many places and apparently is broken by thrust faults. The Mississippian rocks occur at the surface along this anticline as alternating broad and narrow masses and are complexly contorted in contrast with the more uniform structure of the Pennsylvanian rocks on either limb.

The following table gives the succession and character of the strata present at Malagash:

Pennsylvanian	Pictou series	Red shale, sandstone, and conglomerate
	Unconformity	
	Boss Point formation	Massive red, gray, to buff sandstone, with interbedded red and some gray shale
Mississippian	Windsor series	Red shale, sandstone, soft gray shale, interbeds of gypsum and limestone
	Windsor series (?)	Salt and interbedded gray and red shale

These formations are concealed beneath about 25 ft. of boulder clay except along the shore at low tide, as indicated by Fig. 1, which shows the relative position of outcrops to the salt of the mine workings. The levels in this figure serve as structure contours of the Lucas salt seam, which is one of the purer beds of salt in the upper part of the salt strata.

The salt strata explored by the mine workings and by drill holes have a thickness of about 300 ft. and consist of pure salt interstratified with impure salt, and red and gray shale impregnated with salt. Salt, making

¹ The following references furnish further information regarding the geology:

H. Fletcher: Malagash Sheet No. 60; Geol. Survey of Canada.

A. O. Hayes: The Malagash Salt Deposit, Cumberland County, N. S. Geol. Survey of Canada. *Mem.* 121 (1920).

W. A. Bell: Carboniferous Formations of Northumberland Strait, Nova Scotia. Geol. Survey of Canada *Sum. Rept.* (1924) pt. C, 142-180.

G. W. H. Norman: Salt Deposits of Nova Scotia and New Brunswick. Geol. Survey of Canada *Sum. Rept.* (1931) pt. D, 28-35.

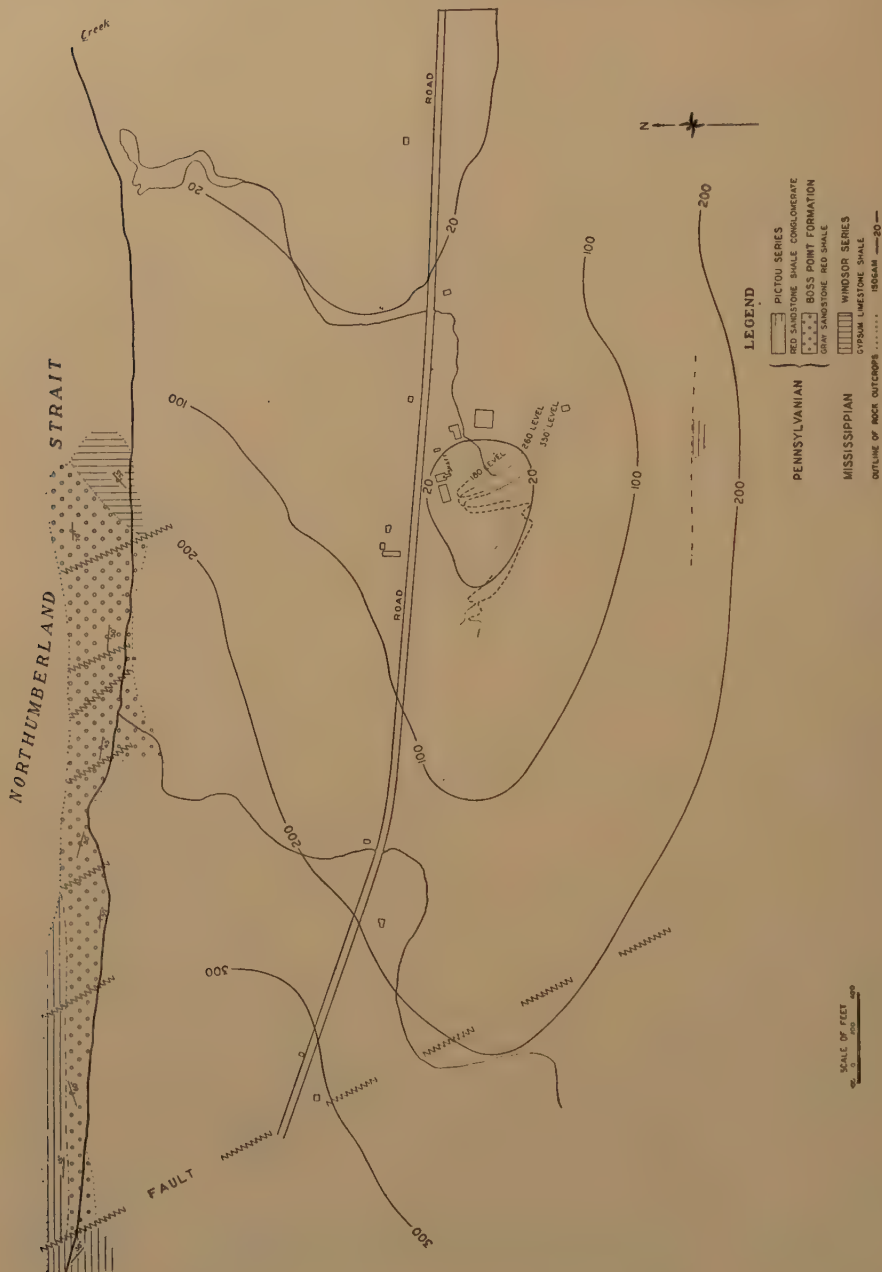


FIG. 1.—PLAN SHOWING GEOLOGY IN THE VICINITY OF THE SALT DEPOSIT AT MALAGASH, N. S.

allowances for clay in the impure beds, amounts to about 65 per cent or so of the total thickness. The character of about 1000 ft. of strata that lie on either side of the salt is known very imperfectly from a few shallow drill holes that were drilled along and near the salt. The holes drilled indicate that the unexposed strata on either side of the salt include red shale and sandstone, gypsum and gypsiferous clays. It is possible that they are a part of the Windsor series that is poorly exposed on the shore $\frac{1}{3}$ mile north of the shaft. The Boss Point formation outcrops only along the shore and is overturned toward the north. The Boss Point strata rest with little if any discordance on the Windsor strata in the Cumberland Basin and at Malagash their contact is possibly along a fault. The Pictou strata have gentle north dips on the shore north of the mine and rest unconformably on the upturned edges of the Boss Point formation. Although the Pictou strata are not exposed immediately south of the mine, except for one doubtful outcrop, they are inferred to underlie a prominent ridge that lies about 1200 ft. south of the shaft, and they outcrop with steep south dips in a westward extension of this ridge.

The salt strata at Malagash probably form part of a domelike structure on a faulted anticline that strikes east. That possibly all or at least the more southerly and uppermost salt beds in the mine are a part of the south limb of the structure is suggested by the consecutive order from north to south of anhydrite, salt and potash-bearing salt that occurs in a small thickness of the uppermost beds. The north limb of the structure, which is partly exposed on the shore (Fig. 1) beneath the gently dipping Pictou strata, is overturned and is probably broken by thrust faults, as the distance from the base of the Boss Point formation to the salt is insufficient to include the thickness of strata that occurs in this interval in other parts of the Cumberland Basin. The salt strata are contorted into steeply plunging folds whose axial planes strike across the anticline, and parallel to a series of north-northwest faults in the more competent strata on the shore. The majority of these transverse faults on the shore are small, but one that lies about 3200 ft. west of the shaft has a downthrow of 500 ft. or more on its west side. The most easterly of the cross faults on the shore should pass a short distance east of the shaft if it parallels the other faults. But as the outcrops along this part of the shore end about 400 ft. east of this fault, its magnitude and the probable location of salt east of it are unknown.

THE GRAVIMETRIC SURVEY

Density Measurements

Before the survey was undertaken 27 samples of rocks including salt were collected at Malagash for density tests, the results of which were as follows:

FORMATION	ROCK	NUMBER TESTED	RANGE IN DENSITY	MEAN
Windsor	Salt	5	2.14-2.24	2.16
Windsor	Impure salt	3	2.16-2.21	2.18
Windsor	Gray clay	1		2.15
Windsor	Yellow shale	2	2.17-2.30	2.23
Windsor	Red shale	1		2.50
Windsor	Sandstone	2	2.32-2.61	2.48
Windsor	Gypsum			2.30 ^a
Boss Point	Red shale	1		2.56
Boss Point	Sandstone	5	2.25-2.45	2.33
Pictou	Sandstone	5	2.32-2.67	2.45
Pictou	Conglomerate	2	2.35-2.38	2.36

^a This value was taken from the Smithsonian Physical Tables.

The measurements indicated that the density of the rocks is rather low but that on the average it exceeds that of salt by 0.2. Some of the rock samples have very nearly the same density as the salt but these, with the exception of Windsor gray clay from the shore $\frac{1}{3}$ mile north of the shaft, were taken from the salt-bearing shales that are interstratified with the salt in the mine.

Observations

The survey was carried out for part of the time with two torsion balances but for the most part only the photographically recording instrument was used. In all 233 stations were established, 36 of these being observed with the visual instrument. The two instruments were compared at the same station at Ottawa before they were taken to the field and also at one of the field stations. The agreement in both tests was entirely satisfactory.

The distribution of stations over the area was not uniform (Fig. 2). This was partly due to the nature of the topography and partly because wooded areas had to be avoided on account of the small staff. Furthermore access to many parts of the area in the early part of the season was impracticable on account of crops. Advantage was taken of these circumstances to test the uniformity of the results by concentration of stations in the vicinity of the mine and north of it. The remarkable uniformity in the results for these stations in particular, and indeed for the results in general, is convincing proof that any irregularities in either the density of the boulder clay or the elevation of the rock surface immediately beneath it must be of such nature that they produce no disturbing effects in the torsion balance results.

Terrain Corrections and Construction of the Isogams

For the purpose of the terrain correction, Jung's modification (in a slightly different form) of Schweydar's formula was used. Levels out to a distance of 60 meters from each station were taken with a 5-in.

Cooke transit with the aid of a metric rod and a tape on which were prominently marked the necessary radial distances in meters, as required by the terrain formula. Use was also made of field books specially prepared for recording the readings and also of special forms prepared for facilitating the computations of the results. For the gradient it was not necessary to carry the computations beyond 60 m. For curvature the terrain correction was computed to 1000 m. For the correction corresponding to topography, between 60 and 1000 m. use was made of a contour map, on a scale of 800 ft. to the inch, contour interval 10 ft. prepared by Dr. A. O. Hayes in 1920². This map was subdivided into a grid of 100-m. squares by north and east lines. The lines were drawn so that one of the grid points (corners of the 100-m. squares) corresponded to the position of the shaft, which served as a reference point. The terrain corrections were computed for a selected number of the grid points only. For obtaining the necessary elevation readings from the map, as required by the terrain formula, a celluloid template was constructed on which the terrain network between the 60 and 1000-m. circles was drawn to scale. After evaluating the two corrections (U_{Δ} and $2U_{xy}$) for the various selected points these values were plotted separately and contours were drawn. The corrections for the various stations were read directly from the two contoured diagrams of the area surveyed.

Some 40 measurements of soil density were made and from these a value of 1.6 was adopted for most of the stations and 1.7 for the rest.

Errors existing in the results of this survey are no doubt due to inaccuracies in the terrain corrections. However, it is not likely that the adopted gradient values for any of the stations is in error by more than 5 Eötvös units. The adopted curvature values are not so accurate. They may all require a considerable constant correction due to distant topography. Relatively to one another they may be as much as 5 E. in error, and possibly more.

In order to compute the relative gravity values, required for constructing the gravity contours or isogams, the coordinate positions of all the stations referred to the shaft as origin were determined from the plan of the survey. The directions of the two coordinate axes were chosen north and east respectively, these directions being also those in which the two components of the gradient were reduced. To obtain the difference in gravity between two stations, for example, it was merely necessary to multiply the computed mean components of the gradients in the two principal directions north and east by the corresponding differences in the coordinates of the stations and to add the two products. Besides being convenient, this method avoided the errors due to scaling and plotting inherent in a graphical method. Often groups of two,

² Reference of footnote 1.

three or four stations were employed. Occasionally the number in a group exceeded four. The position of the group center was obtained by averaging the coordinates of the stations in the group. The components of the gradient for this point were similarly obtained by averaging the corresponding values for the several stations of the group. Altogether 190 values of gravity were computed and plotted on the plan of the survey, although to avoid confusion these are not shown on Fig. 2. Several adjustments were made to compensate for closing errors. The probable error of the adopted values at points where closings were made was never greater than 5×10^{-5} c.g.s.

RESULTS OF THE SURVEY

The results of the survey are illustrated in Fig. 2, which shows the gradients, curvatures and isogams, or lines of equal gravity. The total anomaly measured is about 300×10^{-5} c.g.s., or 0.003 c.g.s.

The isogam contour 20, which is drawn about the shaft, outlines an area of low density under which salt is known to exist. It is most likely, as indicated by computation also, that the gravitational anomalies in the immediate vicinity of the mine are mainly due to the occurrence of salt. The gradients over the known deposit are directed away from the salt, and the curvature is large over it and at right angles to the general strike of the salt beds. The location of the westward extension of the salt from the mine is probably fairly well indicated by the line of maximum curvatures and minimum gradients that extends west from the center of the area of low gravity at the shaft. The results show that an area of low gravity comparable to that over the known deposit occurs east of the shaft. There is only a small rise in gravity over the ground separating the two areas of low gravity. The results over this second area are obviously an indication of salt or other rock of low density. Towards the completion of the survey a shallow drill hole was put down about 100 ft. west of station 215 in the southwestern part of this area and encountered vertical strata of light reddish shale. The densities of three samples from the drill core were found to be almost precisely the same as that of salt. This does not however necessarily preclude the existence of salt in other parts of this area. Whether this area contains a deposit of salt of commercial importance cannot be settled until the area is adequately explored. If salt is present, it probably occurs at about the same depth as at the mine.

An attempt has not been made to compute a hypothetical structure that would fit the observed results. It is felt that even if such a solution were made little could be definitely asserted from the results regarding the minor folds in which the salt lies. The pattern of isogams and gradients over the western three-quarters of Fig. 2 is suggestive of a dome plunging uniformly westward and broken eastward. However, a lens-

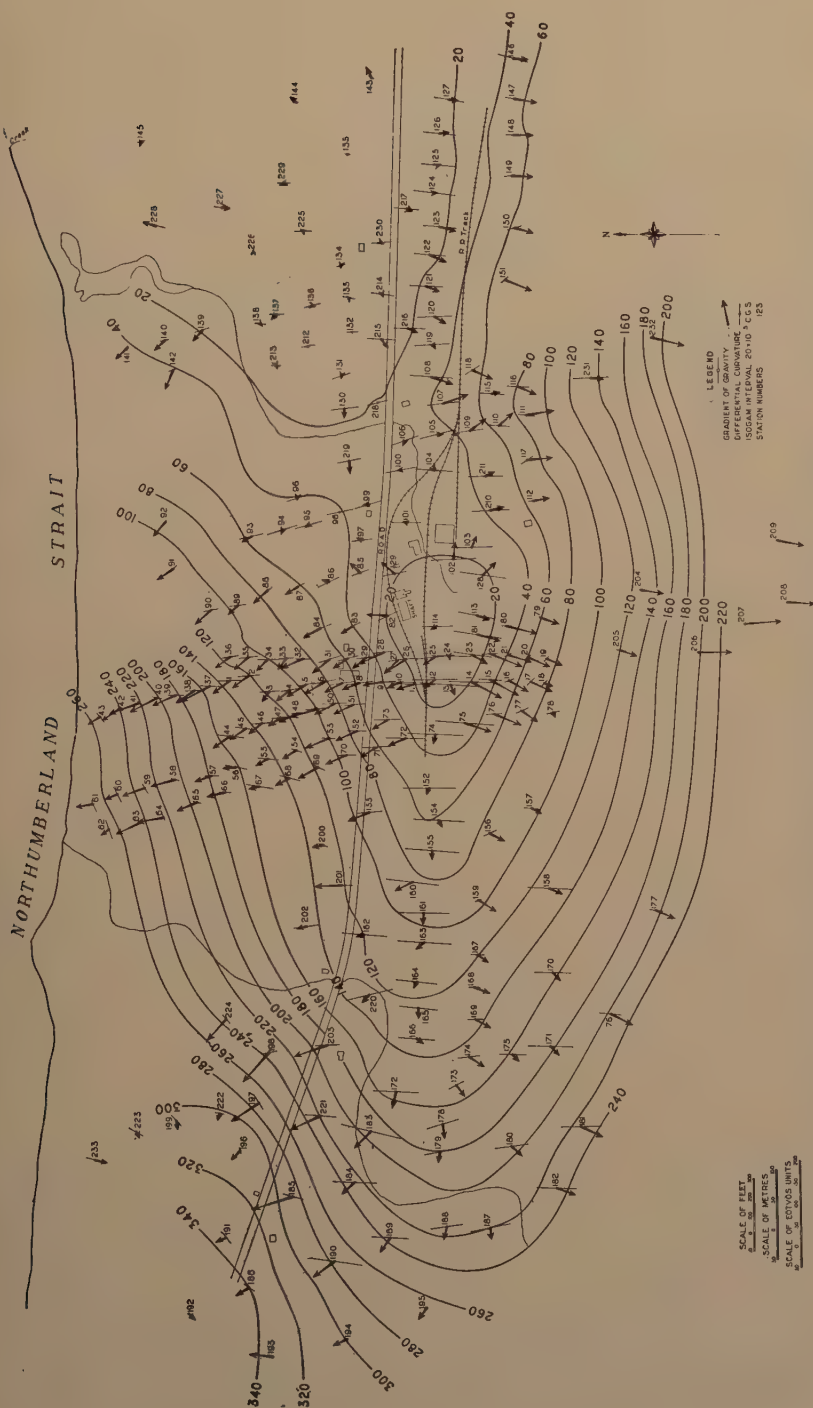


FIG. 2.—GRADIENTS, CURVATURES AND ISOGAMS FROM TORSION BALANCE SURVEY AT MALAGASH, N.S., 1934.

At stations where gradient arrows are not shown, the gradients are too small to be plotted. Curvatures for stations close to the borders of the topographic map used are omitted because the correction for these curvatures could not be computed.

shaped mass of rocks of low density, dipping vertically or nearly so and thinning westward either horizontally or vertically or in both these directions, might produce a similar pattern. Whether the salt is merely folded into its present position or has been forced upwards through younger strata as an intrusive mass cannot of course be determined from the results.

Combined with the more apparent causes of disturbance in the gravitational field there are no doubt other causes that contribute to the general result. Some of these are quite possibly due to subterranean conditions different from those existing near the surface. It seems probable also that the Mississippian rocks taken as a group are of lower density than the Pennsylvanian strata. This is indicated by density measurements and is also suggested by the large gradients and distinct change in direction of curvature that apparently mark the concealed contact of the Windsor and Boss Point formations between the shore north of the mine and the main cross fault to the west of the mine. There are no very apparent gravitational effects associated with any of the cross faults, not even with the largest. The displacements along these faults would seem therefore to involve only minor disturbances of density.

Neither the gradients nor isogams shown in Fig. 2 have been corrected for the salt that has been taken out of the mine. There is doubtless an appreciable effect due to this cause in the vicinity of the mine and this is apparent in some of the observed results. To correct each gradient and curvature thus affected, while not an impossible task, would have been a difficult undertaking, leading in the end to somewhat uncertain results. The computed effect on gravity due to the removal of salt is about 25×10^{-5} c.g.s., or less than 10 per cent of the total anomaly measured. It is thus apparent that the general results illustrated in Fig. 2 would not be materially altered by correction for the gravitational effect due to salt removal.

Mining and Preparation of Rock Salt at the Retsof Mine

BY E. F. LA VIGNE*

(New York Meeting, February, 1936)

ON Aug. 10, 1884, the Retsof Mining Co. began the sinking of an 18 by 12-ft. shaft at Retsof, N. Y. Rock salt was reached in 1885, in September, at 1008 ft. below the surface. The first salt was shipped in the fall of 1885. A timber breaker was built in 1886.

In July, 1895, there was a consolidation of the Retsof Mining Co., the Greigsville Salt & Mining Co., the Livonia Salt Co., and the Lehigh Salt & Mining Co. These consolidated salt companies are now known as the Retsof Mining Co., a subsidiary of the International Salt Co. Eventually all of the mines were closed except the mine of the Retsof Mining Co., at Retsof, Livingston County, New York.

On July 6, 1921, about $\frac{1}{2}$ mile south of the original shaft and plant of the Retsof Mining Co., a new 9 by 28-ft. concrete-lined shaft was started. The shaft is 1063 ft. deep from the top of the shaft collar to the bottom of the salt bed. At the same time a reinforced-concrete breaker and other buildings were erected. This new plant began operations on Dec. 15, 1923. It was designed for a capacity of 3000 tons in 8 hours.

GEOLOGY

The workable rock-salt bed is 1063 ft. below the surface and 326 ft. below sea level, and varies in thickness from 9 to 10 ft. The total thickness is greater than that but is contaminated with shale above and below the pure mineral. The characteristics of the formation are shown in Tables 1 and 2. The bed dips uniformly 0.87 per cent to the south.

PROSPECTING

As the salt deposit in New York State is known, very little prospecting has been necessary. Before the sinking of the Retsof No. 1 shaft, a well was drilled to prove the existence of salt. No. 1 shaft was sunk where the well was located. The several salt wells drilled in Livingston County proved generally the existence of salt throughout the county. (See U. S. Geol. Survey *Bull.* 669: Salt Resources of the U. S.)

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On March 29, 1926, drilling was started to find salt at a greater depth. The drill hole was started at the floor of the mine of the Sterling Salt Co. at Cuylerville, N. Y., $3\frac{1}{2}$ miles south of the Retsof mine. The total

TABLE 1.—*Section of Retsof No. 1 Shaft, Showing Different Strata from Surface to Salt Measures^a*

	DEPTH OF STRATA, Ft.	TOTAL DEPTH, Ft.
Shale rock.....	133	133
Lime rock.....	8	141
Shale rock.....	232	373
Lime rock.....	4	377
Shale rock.....	23	400
Lime rock.....	2	402
Corniferous lime rock.....	142	544
Cement rock.....	13	557
Dark rock.....	4	561
Brown and white cement.....	7	568
Sandstone.....	14	582
Gypsum rock.....	4	586
Cement rock.....	26	612
Gypsum rock.....	47	659
Magnesium lime.....	63	722
Cement rock.....	14	736
Blue rock.....	25	761
Cement rock.....	10	771
Blue rock.....	12	783
Top of Helderburg		
Cement rock.....	17	800
Lime cement and sandstone.....	31	831
Cement rock.....	10	841
Lime cement and sandstone.....	15	856
Cement rock.....	6	862
Blue rock.....	18	880
Red rock.....	10	890
Blue rock.....	42	932
Red shale.....	5	937
Blue rock.....	13	950
Lime rock.....	12	962
Mixture salt and shale.....	33	995
First bed of salt.....	20	1015
Lime rock and shale.....	26	1041
Second bed of salt.....	4	1045
Rock.....	2	1047
Third bed of salt.....	58	1105

^a This shaft was sunk in 1884, and the only identification of rock layers available is detailed herein.

depth of the well was 798 ft. below the mine floor. The elevation of the drill platform was minus 500.2 ft., the elevation of the shaft collar was 617, and the elevation of the bottom of the well was minus 1298 ft.

(U. S. Geological Survey bench mark was used.) The log of the borehole is shown in Table 3.

TABLE 2.—*Log of Fuller Salt Shaft*

	DEPTH OF STRATA, Ft.	TOTAL DEPTH Ft.
Laminated clay.....	9	9
Blue clay.....	7	16
Sandy clay and loose stone.....	3.5	19.5
<i>Moscow Shale:</i>		
1. Soft shale.....	6.4	25.9
Fossiliferous hard gray shale.....	16.1	42
<i>Tichenor Limestone:</i>		
2. Fossiliferous soft gray shale.....	8	50
<i>Ludlowville, Skaneateles and Cardiff Shale:</i>		
3. Fossiliferous soft gray shale and limestone.....	121.3	171.3
4. Fossiliferous soft gray shale.....	4	175.3
5. Fossiliferous gray limestone and barren black shale.....	104	279.3
6. Gray sandy limestone.....	14	293.3
7. Fine-grained lime shale.....	120	413.3
<i>Stafford Limestone:</i>		
8. Black limestone with tiny brachiopods.....	2	415.3
<i>Marcellus Shale:</i>		
9. Black shale with iron pyrites.....	35	450.3
<i>Onondaga Limestone:</i>		
10. Fossiliferous gray to brown limestone.....	49.7	500
11. Gray limestone and chert.....	56	556
12. Chert.....	34	590
13. Gray limestone and chert.....	4	594
<i>Cobleskill-Akron Dolomite:</i>		
14. Dirty, oily brown limestone with tiny crystals...	12	606
15. Dirty, oily brown limestone with tiny crystals...	6	612
16. Porous oily brown limestone with tiny crystals..	8	620
<i>Bertic 46-ft. Waterlime (same Camillus):</i>		
17. Impure gray limestone.....	5	625
18. Gray limestone with splintery fracture.....	24	649
19. Platy light and dark gray limestone.....	17	666
<i>Camillus Shales:</i>		
20. Gypseous shale to mottled gypsum.....	94	760
21. Brown limestone with tiny gypsum crystals....	1.5	761.5
22. Dark gray shale (reported lime and gypsum)...	24.5	786
23. Brown-gray limestone, laminated (looks gyp- seous).....	8	794
24. Soft gray limestone with conchoidal fracture...	4	798
25. Mottled brown to translucent gypsum.....	6	804
26. Gray salt and pepper sandy limestone and edgewise conglomerate.....	2	806
27. Black limestone with conchoidal fracture.....	36	842
28. Brown limestone.....	40	882
29. Gray limestone with irregular fracture.....	8	890
30. Dark shaly limestone.....	5	895
31. Gray limy shale.....	5	900

TABLE 2.—(Continued)

	DEPTH OF STRATA, Ft.	TOTAL DEPTH, Ft.
32. Fine-grained limestone, round cavities and salt and clay.....	1.5	901.5
33. Laminated limy shale.....	7.5	909
34. Dark gray pure gypsum.....	1	910
35. Gray shale.....	4	914
36. Mottled brown limestone, gypsum and anhydrite.....	6	920
37. Gray shale, massive gypsum and red salt beds..	24	944
<i>Vernon Shales:</i>		
38. Red and green shale with pink salt veins.....	8	952
39. Gray limestone with conchoidal fracture.....	7	959
40. Light gray shale with red salt in cavities.....	17	976
41. Green shale with gypsum nodules and pink salt streaks.....	2	978
42. Soft gray shale finely laminated in varied colors.	20	998
43. Hard dark shale.....	1	999
44. Mottled red and gray shale with massive gypsum nodules.....	6	1005
45. Gray and tan banded shale and anhydrite....	3	1008
46. Gray shale with pink salt and a little anhydrite..	12	1020
47. Mottled limestone and salt (solution will leave holes).....	9	1029
48. Gray shale with red salt.....	27	1056
49. Gray rock salt. (This is mined).....	18	1074
50, 51. Gray limestone and shale with red salt beds	62	1136

Table 3.—Log of Bore Hole begun on March 29, 1926

MINE FLOOR	DEPTH OF STRATA, Ft.	ELEVATION, Ft.
Camillus-type magnesian lime mud rock.....	14.7	514.7
Camillus-type magnesian lime mud rock with salt.....	21.0	535.7
Rock salt.....	5.0	540.7
Magnesian lime mud rock.....	4.0	544.7
Magnesian lime mud rock with anhydrite.....	7.0	551.7
Rock salt.....	10.0	561.7
Camillus-type magnesian lime mud rock.....	1.0	562.7
Rock salt.....	16.0	578.7
Camillus-type magnesian lime mud rock.....	1.6	580.3
Rock salt.....	13.4	593.7
Camillus-type magnesian lime mud rock.....	2.5	596.2
Rock salt.....	11.8	608.0
Vernon-type green and black shale.....	4.2	612.2
Rock salt.....	0.8	613.0
Vernon-type green and black shale.....	2.4	615.4
Camillus-Vernon magnesian lime mud rock green shale.....	2.7	618.1
Camillus-type magnesian lime mud rock.....	6.1	624.2
Magnesian lime mud rock black shale.....	1.5	625.7
Magnesian lime mud rock black shale with anhydrite.....	1.0	626.7
Vernon-type black shale.....	2.0	628.7
Camillus-type magnesian lime mud rock with anhydrite.....	9.0	637.7
Camillus-type magnesian lime mud rock.....	1.6	639.3

TABLE 3.—(Continued)

MINE FLOOR	DEPTH OF STRATA, FT.	ELEVATION, FT.
Camillus-type magnesian lime mud rock.....	2.1	641.4
Vernon-type green shale.....	1.0	642.4
Camillus-type magnesian lime mud rock.....	2.8	645.2
Camillus-type magnesian lime mud rock with anhydrite.....	7.6	652.8
Vernon-type red and green shale.....	2.5	655.3
Camillus-type massive anhydrite.....	1.1	656.4
Vernon-type green shale.....	3.7	660.1
Vernon-Camillus red shale magnesian lime mud rock.....	1.2	661.3
Vernon-type green shale.....	1.4	662.7
Vernon-type red shale.....	3.2	665.9
Vernon-type red and green shale.....	4.8	670.7
Camillus-Vernon magnesian lime mud rock red shale.....	1.4	672.1
Camillus-Vernon rock red shale.....	2.4	674.5
Magnesian lime mud rock.....	7.5	682.0
Massive anhydrite.....	1.5	683.5
Red-shale magnesian lime mud rock with anhydrite.....	1.9	685.4
Red-green shale magnesian lime mud rock with anhydrite...	5.6	691.0
Magnesian lime mud rock.....	4.5	695.5
Magnesian lime mud rock with anhydrite.....	4.2	699.7
Magnesian lime mud rock.....	2.5	702.2
Vernon Lockport black shale upper Eurypterid horizon.....	3.7	705.9
Camillus-type magnesian lime mud rock.....	2.8	708.7
Vernon-type green shale and anhydrite.....	16.3	725.0
Magnesian lime mud rock green and red shale anhydrite...	5.2	730.2
Magnesian lime mud rock green and red shale.....	11.0	741.2
Vernon Lockport black shale.....	1.3	742.5
Camillus-Vernon magnesian lime mud rock green and red shale	4.2	746.7
Vernon-type red shale.....	2.6	749.3
Vernon-type red and green shale.....	6.0	755.3
Camillus-type magnesian lime mud rock.....	27.7	783.0
Guelf dolomite.....	102.3	885.3
Eramosa beds, thin even-bedded arenaceous dolomite.....	20.4	905.7
Lockport massive dolomite.....	53.6	959.3
Decen argillaceous dolomite limestone.....	3.0	962.3
Gatis limestone.....	6.0	968.3
Rochester gray limey shale.....	74.0	1042.3
Irondequoit coarse-grained shaly limestone.....	53.9	1096.2
Williamson thin-bedded black or greenish shale.....	7.5	1103.7
Sodus gray and purple shale (limestone with fossils).....	21.5	1125.2
(Gun River) Bear Creek, shale Zygodolba Anticostiensis zone.	6.5	1131.7
Reynales blue-gray limestone.....	15.5	1147.2
Maplewood green shale.....	5.7	1152.9
Thorold gray shale.....	6.8	1159.7
Medine purplish red sandstone and shale.....	100.3	1260.0
Queenston purplish red shale.....	38.0	1298.0

A small quantity of gas and black water was found at elevation —783.7 ft. to elevation —900 ft. The quantity of gas was estimated at 20,000 cu. ft. for 24-hr. flow. The quantity of water encountered was just enough to drill with.

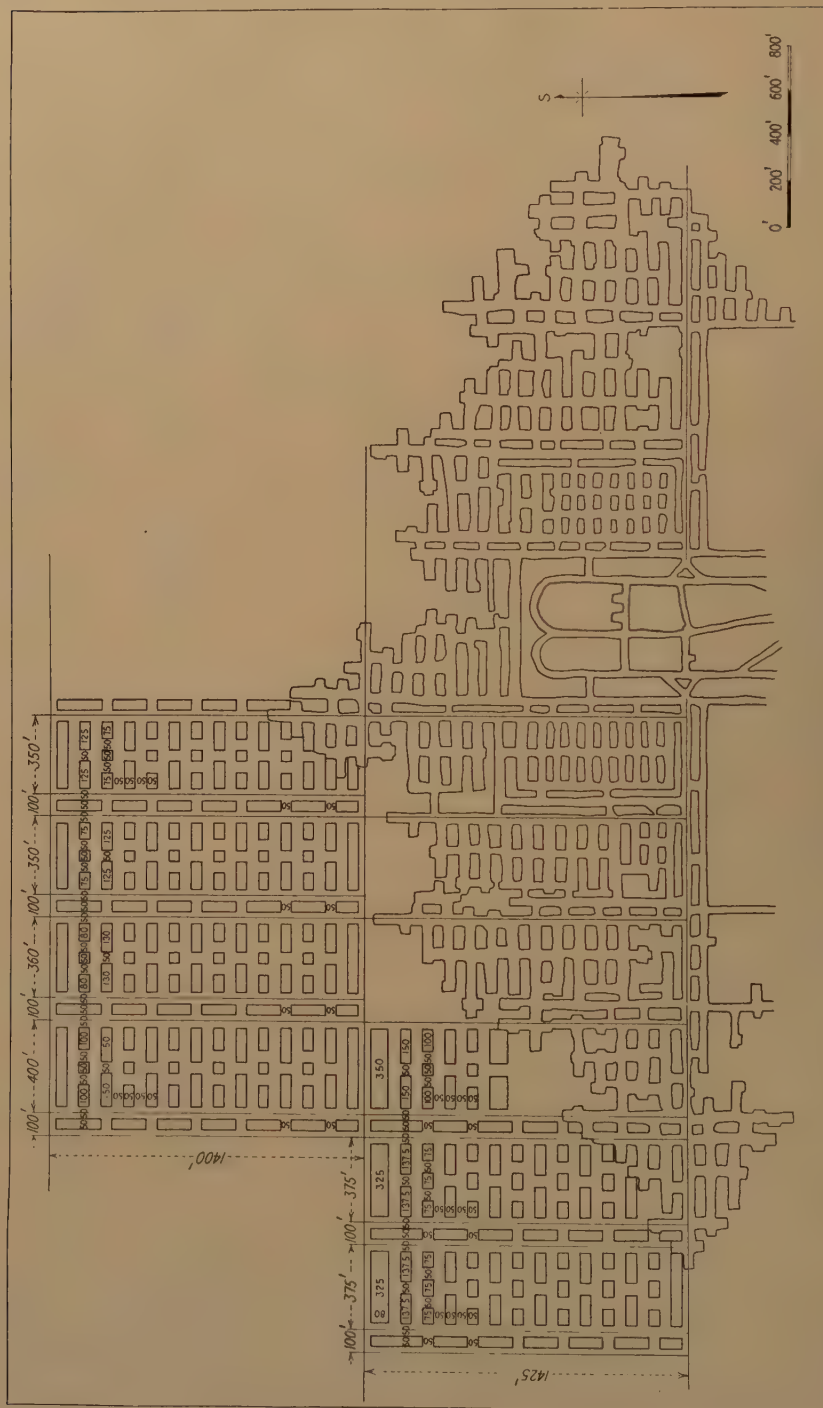


FIG. 1.—PANEL SYSTEM OF MINING, RETSOV MINE.

MINING

When the mine was opened, in the year 1885, a heading 30 ft. wide was driven both ways from the shaft, and irregular pillars were left along both sides of the heading. On each side of this main heading another gangway was driven 30 ft. wide. Off the two gangways, rooms were mined out, 30 ft. wide on 60-ft. centers. Crosscuts were driven 30 ft. wide on 60-ft. centers. The recovery of salt was 75 per cent and 25 per cent was left in pillars to support the roof. This method was followed throughout the mine until 1920.

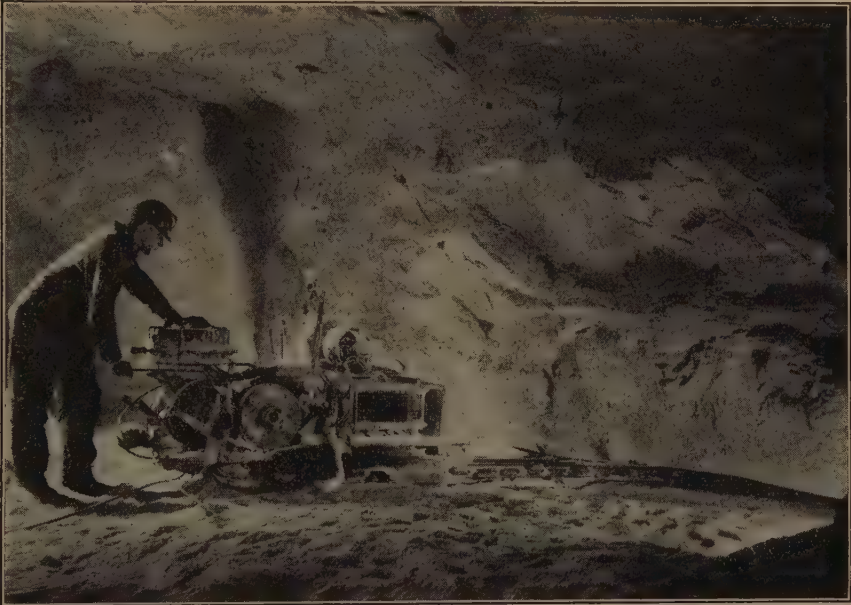


FIG. 2.—UNDERCUTTING. THE HORIZONTAL SLOT AT THE LEVEL OF THE FLOOR IS 55 FEET ACROSS.

When the present shaft, the Fuller, was sunk in 1920, the method of mining was changed to the panel system (Fig. 1). Each panel is 1330 ft. long and 375 ft. wide on centers. Double gangways are driven, one for an airway and one for a gangway. Off the gangway, rooms 50 ft. wide are opened on 100-ft. centers, leaving 50-ft. pillars for roof support. Crosscuts are driven 50 ft. wide; pillars are 50 ft. wide by 112 ft. long. The two end pillars of each panel are not cut through except when necessary to control fresh air. The rooms are driven to a dead end with a crosscut 50 ft. wide but not opened into the adjoining panel except for ventilation. When a panel is mined out it is bratticed off from the mine workings.

Undercutting.—The salt is undercut before drilling, six short-wall machines with a 9-ft. cutting bar being used (Fig. 2). The undercutters are driven by a 50-hp., 250-volt, direct current motor and the cutting speed across the face is 6 in. per minute. The average footage cut by a machine is 120 to 130 ft. per day. The undercutter bits are made from $\frac{3}{8}$ by $\frac{7}{8}$ -in., 18-point tungsten high-speed steel. The bits are drawn, tempered in a coke furnace and ground on a grinding wheel.

Drilling.—From fifty to sixty $1\frac{3}{4}$ -in. holes, 4 to 10 ft. deep, are drilled into the salt face above the undercut. The position of the holes has been

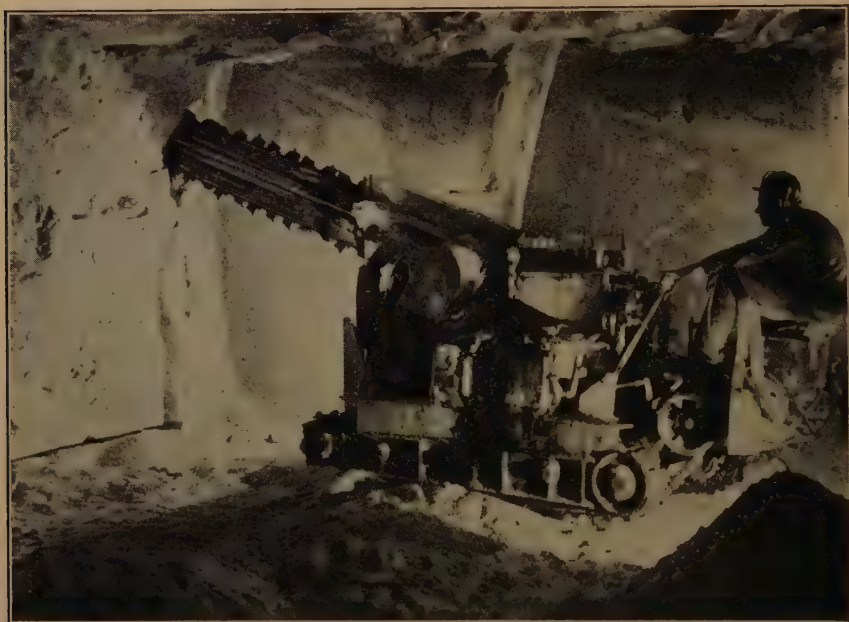


FIG. 3.—SIDE SHEARING. A VERTICAL SLOT IS CUT 9 FEET DEEP FROM TOP TO BOTTOM OF SALT.

determined by long experience. Fig. 4 shows their arrangement and the depth to which they are drilled. The machines used for drilling are the auger-type electric rotary drill with a 3-hp., 250-volt, direct current motor (Fig. 5). The drill steel is $\frac{1}{2}$ by $1\frac{1}{2}$ -in. twisted, nine turns per foot. Special removable cutting heads, of high-speed tungsten tool steel, attached to the auger are used to drill the $1\frac{3}{4}$ -in. powder holes. The feed of the drills is 3 ft. per minute.

Two men make up a drill crew, both experienced drillers. Four set-ups are required to drill a 50-ft. face completely. The drill is supported on a 3-in. extra heavy pipe column, wedged at the roof and the floor of the mine. The two-man drilling crew has been found to be the most economical. Each crew is furnished with a light car, two drill

columns, two drills, drill tables (to stand on when drilling holes that cannot be reached from the floor) duplex covered feed cable, wrenches, bits, picks, shovels and other small tools. When the drillers move into a room to drill a face, the first operation is to set up the two drill columns

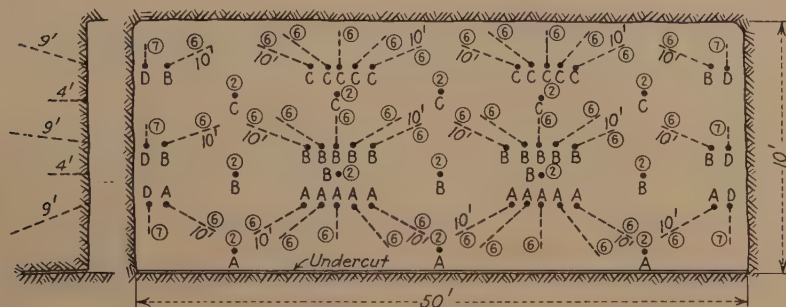


FIG. 4.—METHOD OF DRILLING AND BLASTING.

Shooting sequence. A holes are blasted first, followed by B holes same evening. On second evening, C holes are blasted followed by D holes. Numbers in circles indicate sticks of powder in each hole.



FIG. 5.—MECHANICAL DRILLS, ANCHORED TO FLOOR AND CEILING.

and drills. The first set-up is near the rib. Each driller acts as a helper for the other. About six holes are drilled from the first set-up, then the drill is moved to drill the holes A, B and C indicated on the sketch. About $\frac{3}{4}$ ton of salt per foot of drilled holes is produced. About 19 tons

of salt is drilled per kilowatt-hour. A drill crew drills two faces a day, or 100 lineal feet of room face.

Blasting.—Permissible ammonia extra F. 20 per cent dynamite is used in $1\frac{1}{8}$ by 8-in. sticks for all face drilling. For chunks too big for the shovel to load, a $\frac{7}{8}$ by 8-in. stick of dynamite, of the same strength used in blasting from the solid, is used. All holes are tamped with 18-in. long tamping bags filled with fine powdered salt. No. 6 electric detonators with aluminum shells and 8-ft. iron lead wires explode the powder. The powdermen that work on the night shift load the holes, prepare the tamping, connect up the holes and shoot down the salt. No. 14 insulated copper wire is carried from fixed supports to the face, and conveniently located with reference to the shot-firing station.

The shooting for the entire mine is done by three powdermen, who go into the mine at 3:30 p.m. and finish their shift at approximately 1:00 a.m. In commencing the shift, these men examine the rooms shot the previous day. If all holes have fired and there were no misses, they load the rooms drilled. The powder is stored in a fireproof powder box, kept locked at all times. At each powder station there is a screen to sieve the sawdust from the dynamite. The dynamite is carried into the room in a special carrying box and detonators are carried to the face in a special carrying case.

The number of sticks of dynamite per hole varies from two and one-half to seven, depending on the location and depth of the hole. Generally one to five sticks of powder are put into the drilled hole and tamped lightly with a tamping bar. The second to sixth cartridges are end-primed and put in the hole with the priming end against the powder that was first loaded. Then another stick of dynamite is put into the hole and lightly tamped. Two 18-in. salt-filled tamping bags are then rammed tightly into the hole on top of the powder. The wires are connected in series, using a power current of 250 volts to fire the shot.

Underground Loading.—The broken salt is loaded into mine cars by electrohydraulic power shovels. There are nine shovels in the mine, seven being used to keep up production and the other two as spares and for breakdown service.

The shovels have a capacity of about $\frac{1}{2}$ cu. yd. ($\frac{1}{2}$ ton of coal), full swinging 360° through a complete circle, a 25-hp., 250-volt, direct current motor, which drives a triplex hydraulic pump and also actuates directly two of the mechanical movements of the machine. The hydraulic operations are: (1) jack plunger to roof—raised, held and lowered; (2) scoop thrust forward and back; (3) scoop lifted and lowered; (4) ejector to discharge load from scoop. The mechanical operations direct from the motor are: (5) scoop swing; (6) travel of shovel from place to place on caterpillar treads. The hydraulically operated motions work on an 800-lb. pressure.

A shovel crew consists of the man that operates the shovel; a motor-man, who shifts the loaded cars and keeps the shovel supplied with empty cars; and a brakeman, who also is a helper for the shovel operator.

Two to three cars are loaded before they are shifted from the room. The shovel will load on a 26-ft. radius, therefore one track is laid in the center of the 50-ft. rooms.

The daily average capacity per shovel working an 8-hr. shift is 350 tons. Power lines are carried to the shovel by cable. About 0.5 kw-hr. is required to load one ton of salt.

Percentage of Extraction.—Sixty-three per cent of the rock salt is extracted and 37 per cent is left in pillars. There is practically no loss in the salt extracted, as the approximate amount of rock picked out is 10 tons per week.

Haulage.—All tracks are 36-in. gage, laid with 40-lb. rail; wood and steel ties are used. Main-line haulage tracks are bonded with 0000 copper and electric-welded bonding wire; cross-bond 0000 copper every 100 ft. Refuse and dirty salt is used for track ballast.

The mine cars hold 5 tons without topping, have an open top and welded steel plate construction on a steel frame of solid box type. The top of the mine car is 3 ft. 10 in. high above the top of the rail, 6 ft. 2 in. wide, 8 ft. 6 in. long and 2 ft. 1½ in. deep inside. The axle is 3¼ in., heat-treated; the wheels 16 in. with Timken roller bearings. Couplings are swivel type, to allow dumping without uncoupling the cars of a train.

All hauling is done with electric trolley locomotives, all on 250 volts, direct current. There are 12 locomotives in operation, 8 being used as gathering motors, 2 for main-line haulage and 2 for extra equipment. 4/0 trolley wire is used throughout the mine. A 500,000 and 1,000,000 circular mill cable parallels the trolley wire as feeder.

TABLE 4.—*List of Electric Locomotives*

No.	Size, Ton	Speed, Miles per Hr.	Hp.	Number of Motors	Reel Motor
4	4	5.4	20	2	3
4	5	6.4	26	2	3
2	6	3.2	40	2	3
2	7	6.3	38	2	

Total of 654 hp. on electric locomotives.

Ten of the electric locomotives are equipped with electric gathering reel, driven by a 3-hp. motor and equipped with 250 ft. of No. 2 B. & S. gage rubber-covered cable. On the end of the cable is a hook that fastens on the trolley wire to allow the locomotives to travel to the working face. One-reel locomotive takes care of the tonnage of one shovel—300 to 350 tons per day or shift.

The maximum haul from the working face to the shaft is about $\frac{1}{2}$ mile, minimum haul about $\frac{1}{4}$ mile.

The 0000 trolley wire is hung by $\frac{1}{2}$ -in. dia. rods and expansion shields drilled into the roof. The supporting rods are threaded and cut to the length required, which varies from 6 in. to 3 ft., depending on the height of the room from the track. The trolley wire is kept at a height of 6 ft. 6 in. from the top of the rail. About 0.53 kw-hr. of power per ton is used for transportation.

The maximum grade on the haulage track is 1.5 per cent for a length of about 250 ft. Generally throughout the mine the locomotives when loaded have about 0.5 per cent grade against the load.

Car Dump.—Loaded cars are brought to the shaft bottom in trains of 15 to 20 cars by 7-ton electric locomotives, and are dumped by a rotary car dumper without being uncoupled. The cars are fed into the dumper by an electric reciprocating feeder driven by a 15-hp., 250-volt motor. When a car has been dumped and righted, the operator moves the whole train ahead one car's length, the empty car being replaced on the dump by a loaded one. In this way the loaded train is gradually passed through the rotary dumper to the empty side. The empty train of cars is picked up on the empty side by haulage locomotives and taken to the working faces to be reloaded. The dump, of structural steel construction, 10 ft. in diameter, 9 ft. long, is supported on rollers and revolved by means of wire ropes connected to air cylinders.

When a car is dumped the salt passes over grizzly bars, which separate fines from lump. The large lumps, from 8 to 20 in., are put through a 30 by 42-in. jaw crusher with an 8-in. discharge opening, which is belt-driven by a 75-hp., 220-volt, 720-r.p.m. slip-ring motor. The fines pass through the grizzly bars and with the discharge from the crusher are delivered to two structural steel bins with skip-loading gate outlets, one to each skipway. A center bulkhead of steel divides the bins so that one-half of a car is delivered to each bin. One car of salt is hoisted, or 5 tons per skip per trip. The skip's weight acting on a wiping arm opens the bottom-hinged chute-discharge gate. The gate is brought back in its closed position by counterweights as soon as the skip is hoisted.

Mine Drainage.—The only water entering the mine is seepage coming down the shafts. This water is pumped into one sump by a 3 by 3 triplex and a 3 by 4 triplex pump, each driven by a 5-hp., 220-volt motor. All the water is pumped to the surface by a 3 $\frac{1}{2}$ by 7-in. vertical triplex pump at the rate of 50 gal. per minute through a 3-in. cast-iron pipe line against a head of 1025 ft., and a gage pressure of 500 lb. The pump is gear-driven by a 30-hp., 220-volt induction motor. Twenty to 25 hr. pumping per week takes care of all of the water.

Ventilation.—The mine is ventilated by an 8-ft. Aerovane fan, located in the mine near the main hoisting shaft. Air is drawn down one of the

shaft openings through the working faces and up the main hoisting shaft. Where necessary brattices made of rock salt are built to control the flow of air. There are three automatic mine doors on the main haulage line that controls and prevent short-circuiting of the air. The ventilation is very simple, owing to the small number of men working, the absence of gas and the small amount of powder used, 0.4 lb. per ton of salt. The fan is driven by a Texrope drive using a 25-hp., 250-volt motor, at a speed of 524 r.p.m., delivering 100,000 cu. ft. of air at $1\frac{1}{4}$ -in. water gage.

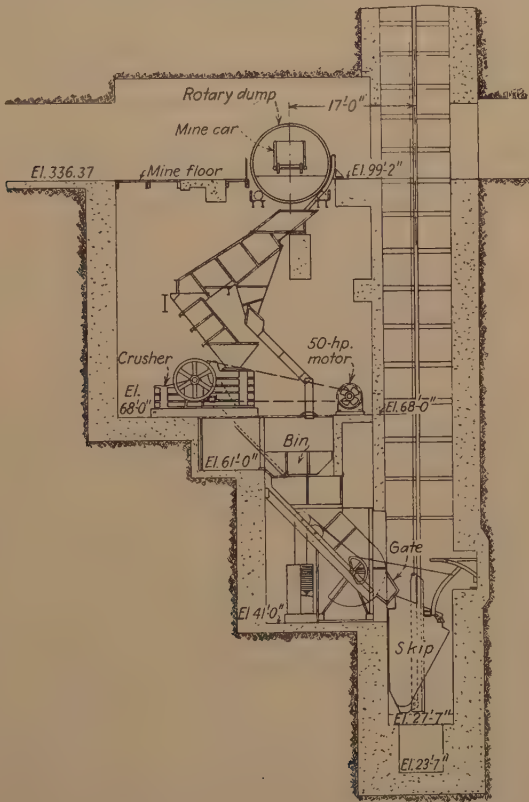


FIG. 6.—SECTION OF FULLER SHAFT BOTTOM.

Shaft.—The shaft (Fig. 6) is concrete lined, 9 ft. wide by 28 ft. long, with circular ends, and the distance from the top of the shaft collar to the bottom of the sump is 1150 ft. The concrete lining varies in thickness from 12 in. at the ends to 18 in. at the center of the shaft. The bottom of the salt bed is 1063 ft. from the surface, the 77 ft. below the bed being required for crushing and for loading the skips. The shaft is divided into three compartments, two for the skips and one for the service or man cage, with a small balance weight compartment for the cage. The $5\frac{1}{2}$ -ton bottom-discharge steel skips operate in balance. Two $1\frac{3}{4}$ -in. dia.,

six-strand, 19-wire plow-steel layset ropes are used. The hoisting cycle is 75 seconds. The balanced service cage is hoisted by a $1\frac{1}{4}$ -in., six-strand, 19-wire plow-steel rope.

Hoist.—The shaft is equipped with an Ottumwa Iron Works cylindroconical drum, with single-reduction herringbone gears enclosed in an oiltight gear case between two sets of bearings, 17 in. in diameter by 33 in. long. The drum is cylindroconical, the small diameter being 96 in., the larger 144 in. The braking system is oil, which consists of an accumulator, automatically controlling the oil pumps and maintaining a continuous supply of oil under constant pressure. The brake shoe is 96-in. dia. by 12-in. face lined with friction wood. The hoist weighs about 70,000 lb. It is driven by a 1050-hp., 375-r.p.m., 2200-volt, 25-cycle induction motor. All possible safety devices are provided to prevent too high a speed, overwinding, power interruption, wrong direction of starting and failure of oil supply.

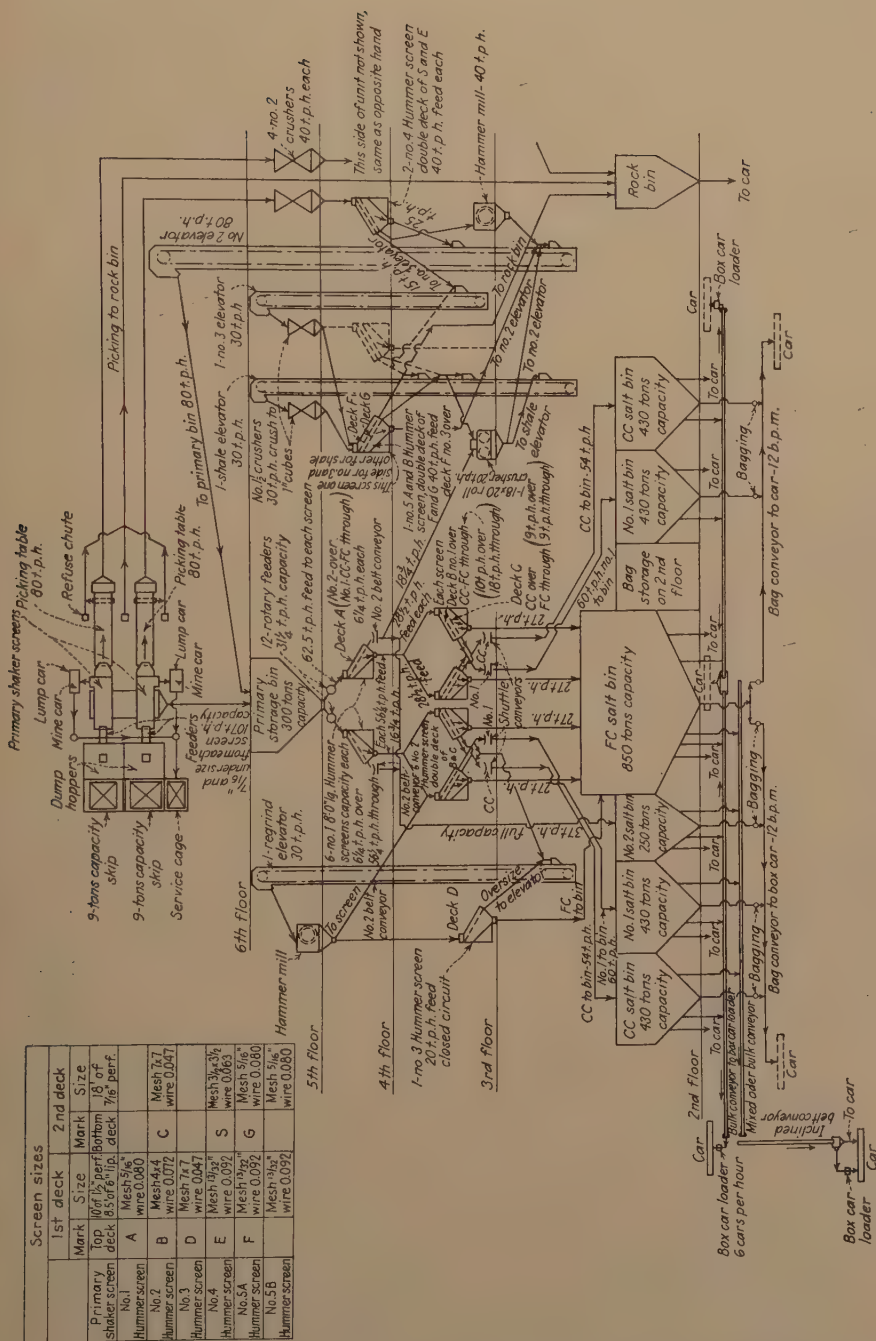
The service hoist has a drum 7 ft. in diameter, 8 ft. 4-in. face, on which is wound two $1\frac{1}{4}$ -in. dia. steel cables. The hoisting speed is 500 ft. per minute. It is driven by a 200-hp., 2200-volt, 25-cycle induction motor. The braking system and safety devices are the same as on the skip hoist.

SCREENING AND CRUSHING

The skips dump directly into a structural-steel dump hopper with a bulkhead in the center, so that the discharges from the skips are separated. In the bottom of each part of the hopper is an opening 4 ft. 3 in. long by 2 ft. wide through which the salt is fed to two double-deck shaking screens by two reciprocating feeders, 5-in. throw, 44 strokes per minute. The feeders are belt-driven by two 20-hp., 250-volt, slip-ring motors.

There are two double-deck shaker screens 4 ft. 7 in. wide by 28 ft. long. The upper screen has a 4-ft. blank plate where the discharge from the feeder falls on the screen, 10 ft. of $1\frac{1}{2}$ wound perforated plate $\frac{1}{4}$ -in. thick, 8 ft. 6 in. of 6-in. lip screen and 7 ft. of blank plate and chute. The lower screen is $\frac{9}{16}$ -in. wound perforated $\frac{1}{4}$ -in. plate. All material passing through this $\frac{9}{16}$ -in. plate is a mixture of all of the sizes of rock salt manufactured. Under the $\frac{9}{16}$ -in. plate is a blank $\frac{1}{4}$ -in. steel plate that chutes the "slack" salt to a 300-ton primary storage bin. The upper and lower deck has a 5-in. throw, 129 strokes per minute, and the screen is set on a 15° pitch.

The top plate of the shaker screen discharges lump rock salt, 6 in. or larger, directly to the picking tables. The shaker screens are operated by an eccentric connecting rod from a counter shaft, driven by the same motor that drives the reciprocating feeders. Lump salt taken from the shaker screens is delivered to special lump cars of steel construction with hinged side gates, and delivered to the lump platform on the surface,



where it is piled in the open and watered, to toughen it. From the lump pile it is loaded, either loose or in bags, to box cars for shipment.

The shaker screens discharge directly to two 48-in. rubber-covered conveyor belts 19 ft. long on centers operating at a speed of 60 ft. per minute and gear-driven by 5-hp., 220-volt motors. The drive pulley of the belt conveyor is a magnetic pulley 18 in. in diameter by 48 in., 220 volts, $3\frac{7}{16}$ -in. dia. shaft, six coils mica insulated from a dynamo steel casting. The coils are protected from mechanical injury by a heavy bronze shield. Rock, blasting wire and refuse are picked from the picking belt by one workman. The two picking belts discharge into four No. 2 open-door, vertical rotary crushers, direct connected to 10-hp., 900-r.p.m., 220-volt, three-phase slip-ring motors by silent chain drives. Each crusher has a rated capacity of 40 tons of rock salt per hour, from 8-in. lump to salt that will pass through a $\frac{3}{4}$ -in. mesh screen.

The discharge from the No. 2 Sturtevant crushers flows by gravity to two No. 4, 30°, 8-ft. single surface electrovibrating screens, oscillated at 900 vibrations per minute by a solenoid operated by 15-cycle, 100-volt alternating current. Each screen has a rated capacity of 80 tons per hour, 24 tons through and 56 tons over. The size of the screen opening is $1\frac{1}{32}$ in. and 0.092-dia. wire.

The salt that goes through the No. 4 vibrating screens is delivered to 12 by 18-ft. vertical bucket elevators, 61-ft. center, gear-driven by 15-hp., 220-volt motors at a speed of 102 ft. per minute. The capacity of each elevator is 80 tons per hour.

The oversize from the No. 4 vibrating screens is delivered to two 8 by 8 vertical bucket elevators, 33 ft. on centers running at a speed of 145 ft. per minute, with a rated capacity of 30 tons per hour each. They discharge to two No. 1½ vertical rotary crushers, direct connected to 5-hp., 220-volt, three-phase slip-ring motors by silent chain drives. These crushers have a capacity of 30 tons per hour, reduce the salt from $\frac{3}{4}$ -in. cube to $\frac{3}{8}$ in. and under.

Salt from these crushers flows to two double-deck electrovibrating screens 8 ft. wide, 30° pitch, with $1\frac{1}{32}$ -in. mesh on the top screen and $\frac{5}{16}$ -in. mesh on the bottom screen. The salt screened through the $\frac{5}{16}$ -in. mesh is chuted to No. 2 elevator, previously described and is delivered to the primary storage bin.

The oversize from the top screen is discharged into one of the No. 3 bucket elevators, previously described, elevated and chuted to two No. 1½ vertical rotary crushers then returned to 8-ft. double-deck electrovibrating screens, 30° pitch. In this closed circuit the hard shale is rejected and discharged to the rock bin (Fig. 7).

The slack salt from the primary bin flows by gravity to six 8-ft. Hum-mer screens, 30° pitch, with single surface $\frac{5}{16}$ -in. screen cloth, 0.080 wire. The oversize No. 2 salt is delivered directly to the storage

bin by two belt conveyors, 18 in. by 46 ft. center to center, driven by 10-hp., 220-volt motors at a speed of 200 ft. per minute, and a rated capacity of 25 tons per hour. The undersize flows, by gravity, to a battery of 12 double electrovibrating screens, each 8 ft. long, 30° pitch, with 4 by 4 mesh, 0.080 wire on top deck; 7 by 7 mesh, 0.047 wire on bottom deck. The salt (No. 1) passing over the 4 by 4 mesh screen discharges to an 18-in. by 105 ft. long reversible self-propelling shuttle conveyor, traveling 200 ft. per minute at a rated capacity of 75 tons per hour, driven by a 15-hp., three-phase, 220-volt synchronous motor that delivers the salt to No. 1 storage bin. The salt (CC) caught on 7 by 7-mesh screen passes to a shuttle conveyor, 18 in. by 105 ft. long, that has a capacity of 75 tons per hour, traveling 200 ft. per minute, driven by a 15-hp., 220-volt, three-phase, induction motor, and is delivered to CC storage bin. The material passing through the 7 by 7-mesh screen discharges directly into the FC salt bin under the screens.

There is one storage bin for No. 2 salt, having a capacity of 250 tons; one FC bin, 800 tons; two No. 1 salt bins, one on each side of the center of the breaker, 400 tons each; two CC bins, approximately 400 tons each. From the bottoms of these bins are chutes that deliver the salt to bulk-storage conveyors and measuring bag chutes.

LOADING

Bulk Loading.—Three conveyors deliver salt from the storage bins to three box-car loaders. One conveyor is 24 in. wide by 134 ft. long and 24 in. wide by 140 ft. long, with troughing idlers, running at a speed of 250 ft. per minute and a capacity of 250 tons per hour; each one gear-driven by a 3-hp., 220-volt squirrel-cage motor and five-ply $\frac{1}{8}$ -in. rubber-covered belt.

Three 24-in. belt-type box-car loaders are used for loading bulk salt. Their capacity is 6 tons per minute each, at a belt speed of 326 ft. per minute. They are chain-driven by 15-hp., 750-r.p.m., three-phase, squirrel-cage open motors. These loaders are supported on roller-bearing brackets and are easily maneuvered for end and center loading of box cars.

The freight cars are dropped by gravity from the empty-car yard to the box-car stations, at each of which is a 50-ft. standard track scale. The salt is loaded in cars to the correct weight and no hand trimming is necessary.

From the storage bins, salt is delivered to measuring chutes, which measure the correct amount of salt for the bags, ranging from 56 to 200 lb. per bag. The salt-measuring chutes are built in double units so that when one side of the hopper is being discharged into bags, the other unit is being filled. The gate of the measuring hopper is operated by air. When a bag is filled it is sewed by an electrically operated sewing machine.

Bag Loading.—Two sewing machines, mounted on trucks that run on tracks underneath the bins, sew all of the salt shipped in bags. From the sewing machines the filled bags are conveyed to box cars. There are two bag conveyors, each 24-in. five-ply rubber-covered belt 118 ft. long, gear-driven by a 5-hp. induction motor at a speed of 250 ft. per minute. These two conveyors run lengthwise of the breaker and parallel to the loading tracks. At each end of the loading conveyors are two cross bag conveyors, 24 in. by 38 ft. long, gear-driven by a 5-hp., 220-volt induction motor, hinged and counterweighted so that they discharge directly into the cars, running at a speed of 250 ft. per minute. Bag salt is piled into box cars by hand.

SAMPLING

Few mine samples are necessary but samples of the various grades of salt are taken daily. Four different sizes are produced: No. 2, No. 1, coarse C and fine C (Table 5). Samples of all grades are taken four

TABLE 5.—*Sizing of Rock Salt*

DESIGNATION OF SALT SIZE	THROUGH, MESH	OVER, MESH
No. 2.....	$\frac{3}{8}$	$\frac{1}{4}$
No. 1.....	$\frac{1}{4}$	$\frac{5}{32}$
CC.....	$\frac{5}{32}$	8
FC.....	8	

times daily at the loading chutes, and are a cross section of the discharge of the chute; 8 to 10 lb. of salt is taken per sample. The samples of each grade of salt are thoroughly mixed and quartered until there remain 1000 grams of each of the four grades.

Sieve analyses are made daily of the four grades of salt. Chemical analyses and physical analyses are made monthly. (Table 6.)

SAFETY METHODS

An active safety organization has been in operation since 1924. A Safety Committee, composed of the heads of the various departments (13 members), meets once a month and at each meeting 10 or 12 workmen attend by invitation. These workmen form a Monthly Inspection Committee, of which the members are changed each month, so as to give each workman an opportunity to be on the committee.

At the monthly meetings all accident records and unsafe practices are discussed. Everyone is encouraged to bring to the attention of the meeting unsafe conditions or practices. After the business meeting has been adjourned, a lunch is served.

An emergency hospital is maintained in charge of a trained nurse. A resident doctor is available at all times.

Inspection is made of the plant by a State Mine Inspector from time to time and his recommendations are carried out.

TABLE 6.—*Analyses, Coarse C Grade*
AVERAGE DAILY SIEVE ANALYSIS, JUNE, 1931

	No. 1 Range Retained on Mesh		CC Range Retained on Mesh		FC Range Retained-on Mesh	
	3½	4	5	6	10	10
Mesh.....	0.223	0.178	0.132	1.04	0.066	0.066
Opening, in.....		3.7	40.1	38.5	15.4	2.3
Monthly average, per cent.....						

CHEMICAL ANALYSIS (AVERAGE), PER CENT

Insoluble matter.....	0.195	Water in soluble matter.....	0.489
Calcium sulfate.....	1.341	Calcium sulfate.....	1.116
Calcium carbonate.....	0.069	Calcium chloride.....	0.079
Calcium chloride.....	0.079	Sodium chloride.....	98.316
Sodium chloride.....	98.316		

PHYSICAL ANALYSIS, PER CENT

Solution Test for June, 1931

GRADE	WATER INSOLUBLES
No. 2.....	1.09
No. 1.....	1.00
CC.....	0.94
FC.....	1.11
Dark salt.....	2.25

Average for May, 1931

	DARK SALT	SHALE
No. 2.....	10.69	0.269
No. 1.....	9.31	0.198
CC.....	4.58	0.084

Some changes and improvements that have been made as a result of the activities of the Safety Committee are:

1. A reduction in lost-time accidents.
2. Use of safety shoes by all men in the mine.
3. Safety hats for all shovel operators.
4. Leather gauntlets for all machine operators.
5. Use of goggles by all men doing work in which flying chips may injure the eyes.
6. Use of carrying cases for powder and fuses.
7. Guarding of all machinery.
8. Use of safety belts on shaft work.
9. Physical examination of all employees.
10. Every minor accident dressed and treated by the nurse or doctor.
11. Rigid enforcement of all safety rules and regulations.

ACKNOWLEDGMENT

The author wishes to acknowledge the assistance given by Mr. T. F. Courthope, General Manager; Mr. Stanley Martin, Superintendent, and others of the Retsof Mining Co., in the preparation of the subject matter and illustrations in this article.

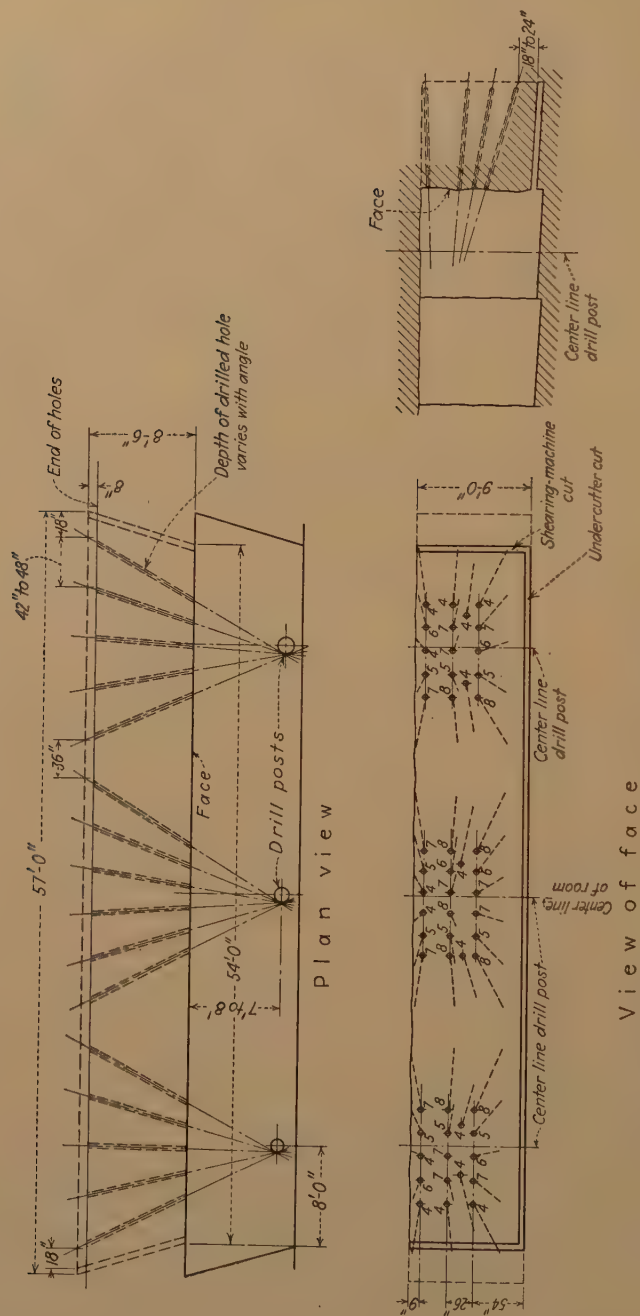


Fig. 8.

DISCUSSION

(H. I. Smith presiding)

G. A. HEATON,* Carlshad, N. M. (written discussion).—The author states that four drill setups are required to drill completely a 50-ft. face. The drilling diagram, however, seems to indicate that seven setups would be required, four for the 10-ft. holes and three for the 4-ft. holes.

The locations of the shear cuts are not shown. Since blasting is a costly item in this type of mining, it would be particularly interesting to know how shear cutting affected: (1) the number and location of holes per face and (2) the total charge per round as compared to a face that is not sheared.

The diagram of the shooting sequence indicated that the second holes from the ribs in the top row are exploded before the short holes in the next row below. Are not the short holes intended to act as "relievers" for the next higher row of holes?

Are the tamping bags filled with salt by the powdermen underground or is some special method of filling these bags adopted?

Having had some experience with electrohydraulic shovels, I am interested to know what type of packing has been found to be most suitable for these units.

The author states that a recovery of 75 per cent was obtained from the original mine while the present extraction is in the neighborhood of 63 per cent. Has any subsidence or breaking of the roof, heaving of the floor, or compression or failure of the pillars been noticed in the old workings? Have any plans been made for mining of the pillars to obtain approximately 100 per cent extraction?

Much experimental work has evidently been done on undercutter bits, since the cutting is so hard. What shape of bit point has been found to give the best results, and what bit arrangement is used in the chain?

T. F. COURTHOPE,† Retsof, N. Y. (written discussion).—Most of the questions Mr. Heaton asks in connection with drilling and shooting are answered graphically on Fig. 8. Shear cuts eliminate *D* holes, shown in Fig. 4, and one setup has reduced powder consumption about 25 per cent.

Shooting sequence is as indicated in the paper and short holes are for the purpose of eliminating secondary shooting of large chunks—not as "relievers."

Tamping bags are filled by undercutter helpers. Packing for electrohydraulic shovels consists of one ring of white cotton fabric on each end of the stuffing box as binder with graphite flax packing as filler.

Heaving of floor and some disintegration of pillars have been noticed in old workings, but there have been no actual failures. No plans are contemplated for robbing pillars.

A tungsten steel bit is used for undercutting, measuring $1\frac{1}{2}$ by $2\frac{7}{8}$ in. and drawn to a cutting point of about $\frac{1}{32}$ -in. radius. Bits are all heat-treated. Type 50 Goodman cutter chains, 7 position, are used. V-type arrangement of bit blocks is found most practical for our work.

W. R. CHEDSEY,‡ State College, Pa.—Why were permissible explosives adopted in a salt mine in which no gases were found?

H. I. SMITH,§ Washington, D. C.—In salt mining the areas at the faces are usually large and the air moves slowly, thus making the relative volume of poisonous gases

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† General Manager, Retsof Mining Co.

‡ Professor of Mining, Pennsylvania State College.

§ Chief, Mineral Leasing Division, U. S. Geological Survey.

given off by explosives an important matter; for instance, black powder gives off many times the volume of carbon monoxide given off by permissible explosives. Another advantage of a permissible explosive is that, when exploded, it does not stain the salt black, as does black powder. Even one black speck in white salt would be very noticeable in a glass salt shaker on your table.

H. J. BROWN,* Boston, Mass.—The author says that a jackhammer is not satisfactory. I think it depends on the type of bit that is used. Ordinarily twist-drilled steel with a Z-bit on the end will drill as fast as a rotary drill. The drill used in the Retsof mines is the Scranton electric drill, the same kind that is used in anthracite mines and in mines producing the harder kind of bituminous coal. The bits used are ordinary fishtail, the steel is twist steel usually six or seven “pods” (turns or twists) per foot. Two men, a driller and a helper are usually employed. But a little jackhammer weighing around 25 or 30 lb. is not difficult for one man to handle in almost any position; by means of horses and a plank he can drill just as fast or even faster.

H. I. SMITH.—I might be able to answer that question in part. The United States Potash Co. at Carlsbad, N. Mex., used jackhammers for about two years. In cutting dry, there was a tremendous amount of dust, and so the cuttings were flushed with water.

H. J. BROWN.—Of course, with salt, water cannot be used.

H. I. SMITH.—Water was used very successfully. The principal objection to it was its effect in increasing the tendency of the salt to hang up in the loading bins at the shaft bottom and storage bins on the surface. The crude potash salt is composed roughly of one-half ordinary salt and one-half potassium chloride. The crude salt contains a small amount of free water and at times this would cause the cuttings to pack when the hole was drilled dry.

H. J. BROWN.—The comment I would make is that probably twist steel and a jackhammer fitted with either a Z-bit or a fishtail properly tempered have not been tried. That combination will drill just as fast and no dust will be produced—cuttings only.

MEMBER.—In salt?

H. J. BROWN.—In salt, yes.

J. R. THOENEN, Washington, D. C.—At the Retsof mine they told me that it was not possible. They have tried the Z-bit and various types with twist steel and a jackhammer.

A. R. CHAMBERS,† New Glasgow, N. S.—I visited the Retsof mine once or twice before its wonderful equipment was revamped. It is certainly a most interesting and instructive place to visit. The economy in production of that mine is wonderful.

We have used both the jackhammer and the Scranton drill that Mr. Brown mentioned, and I will have to confirm what Mr. Brown says about the jackhammer drill. Nevertheless, it has its limitations. The jackhammer drill, we find, will drill a round hole very nicely and quickly, and the smallest size of jackhammer will do pretty good work in rock salt.

* Consulting Engineer.

† President and General Manager, Malagash Salt Co. Ltd.

H. J. BROWN.—One man with a little jackhammer can do a great deal of work, and if he is paid a good, fair wage on the basis of what he produces, and the mine is laid out properly—and I am not sure that undercutting and sidecutting is the most economical way—it seems to me that a good single fast cut, followed by overhand sloping and a second run with the loading machines might cut costs. It is all a matter of conjecture, but it will bear investigation. That is not done so much in nonmetallies; it is hardly done at all outside of some metalliferous mines. The smaller jackhammer is fairly light in weight which makes for rapid shifting from hole to hole and face to face. With proper handling it will run away from any electric drill with which I am acquainted.

H. RIES,* Ithaca, N. Y.—I have been in a good many mines in Europe and in the potash mines in Mexico, and I understand that setup is much more convenient for drilling angular holes. They have a high bed, and you cannot get that setup for drilling a series of holes.

H. J. BROWN.—Of course, care must be taken in picking the jackhammer operators. A small man cannot handle even a light jackhammer, although these jackhammers are not heavy; a man can hold one above his shoulder height and drill holes. No setup is required, hence, no blocking. A pair of horses and one or two heavy planks, as a scaffold allows any placing of holes desired. Of course, 12-ft. or 14-ft. holes cannot be made with a small jackhammer. Such equipment means short holes and a lot of them. Powder expense does not go up—ordinarily 16 per cent salt dynamite. My feeling is that a lighter dynamite, not over 20 per cent, properly loaded in holes efficiently spaced, is the most economical. I do not approve of black powder under any circumstances.

* Professor of Economic Geology, Cornell University.

Evaporating Salt from the World's Largest Mineral Deposit

BY JOSEPH C. BUCHEN*

(New York Meeting, February, 1937)

ABSTRACT†

IN principle, production of salt from sea water is a simple operation. The sun and wind cause evaporation of sea water trapped in ponds, and what is left is principally salt. Commercial production, however, requires consideration of many factors. First is a high consistent annual rate of evaporation, as well as large areas of suitable low-lying ground close to economical shipping facilities. The south end of San Francisco Bay has these requisites. Large ponds are built, by using clamshell dredges to build up levees to a height of approximately 3 ft. on ground usually lower than sea level, so that the water may be taken in through tide gates without pumping. Ponds are divided into extremely large concentrating ponds and smaller crystallizing ponds. Approximately ten times more area is required for concentrating than for crystallizing. The concentrating ponds are irregular in shape and size but the crystallizing ponds are rectangular and fairly regular in size, to aid in harvesting.

The ocean water is taken into the first of a series of concentrating ponds and gradually moved forward from pond to pond as it increases in density, reaching the crystallizing area as a saturated brine. This brine is pumped into the crystallizer at the beginning of the net evaporation season, usually in May, and further evaporation causes its sodium chloride content to deposit on the bottoms of the ponds. The brine must be drawn off and a fresh supply added from time to time to prevent other undesirable salts from depositing with the sodium chloride.

Harvesting usually begins near the end of August and now is accomplished almost entirely by mechanical means. After harvesting, the salt is stored in large stockpiles until required for shipping or refining.

The greatest amount is used for industrial purposes and is shipped without any processing other than screening, but all salt for domestic use produced by the Leslie Salt Co. is refined by the vacuum process, giving a product consistently testing over 99.9 per cent sodium chloride.

* General Superintendent, Leslie Salt Co., San Francisco, California.

† The entire paper was published in *Mining and Metallurgy* (July 1937) **18**, 335.

The Salt Industry of Louisiana and Texas

By W. M. WEIGEL,* MEMBER A.I.M.E.

(New York Meeting, February, 1935)

SALT production in Louisiana and Texas at the present time is entirely from salt domes of the interior and coastal groups. Before and during the Civil War salt was recovered from numerous salines and salt springs by evaporation in open kettles. The location of these in nearly every instance was near what is now known to be a salt dome, and the source of the salt in the salines and spring waters was that in the salt core. The tops of many of the shallower domes have been subjected to erosion¹.

The discovery that the salt occurred as large masses of rock salt in the form of domes was made in May, 1802, at Avery Island, La., by John Marsh Avery, grandson of the original owner, John C. Marsh, who built the old salt works in 1812². The first shaft for mining was sunk on the Avery dome in 1867 by Chouteau and Price. It was 90 ft. deep, about 58 ft. being in solid salt. It was abandoned in 1870.

There are three general groups or fields of salt domes; interior Louisiana in northern Louisiana, interior Texas in northeastern Texas, and the coastal domes nearer the Gulf Coast and extending from the Mississippi River to the Rio Grande. This last group contains by far the greatest number of domes. As many of the coastal domes especially are prolific producers of petroleum and a few of sulfur, their geology has been intensively studied and fully covered in the literature.

The depth to the top of the salt varies from near the surface, where shallow ground waters have dissolved the salt to just below recent deposits, to several thousand feet, where the depth has been estimated only by geophysical methods. The diameter of the salt core varies from less than $\frac{1}{2}$ mile to over 2 miles. Only domes where the salt is comparatively near the surface have been developed for commercial salt. Three deeper domes, which will be described later, have recently been developed or are in process of development, as a source of brine for chemical use.

Fig. 1 shows the location of the domes so far developed for commercial salt and brine. Nos. 1, 2, 4, 7, 8, 9 and 10 are in the Gulf Coastal field,

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¹ S. Powers: Interior Salt Domes of Texas. *Bull. Amer. Assn. Petr. Geol.* (1926) 10, No. 1, 14.

² F. E. Vaughan: The Five Islands, Louisiana. *Bull. Amer. Assn. Petr. Geol.* (1925) 9, No. 4, 762.

while Nos. 3 and 6 are in the interior Texas field and No. 5 in the interior Louisiana field.

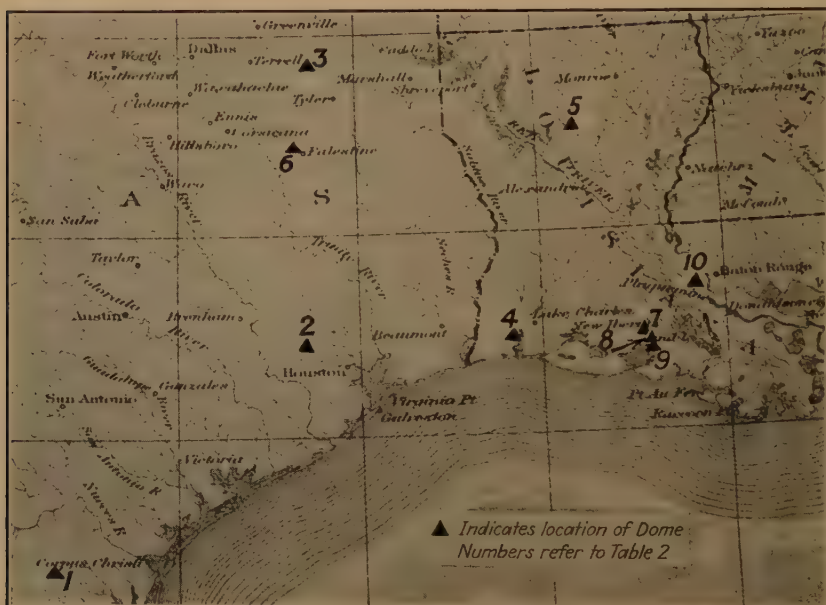


FIG. 1.—MAP OF LOUISIANA AND PART OF TEXAS SHOWING LOCATION OF SALT DOMES DEVELOPED FOR SALT AND BRINE.

Fig. 2 is a cross-section of the Palestine dome³, No. 6. It is fairly typical of the shallower domes except that in most other domes the sides

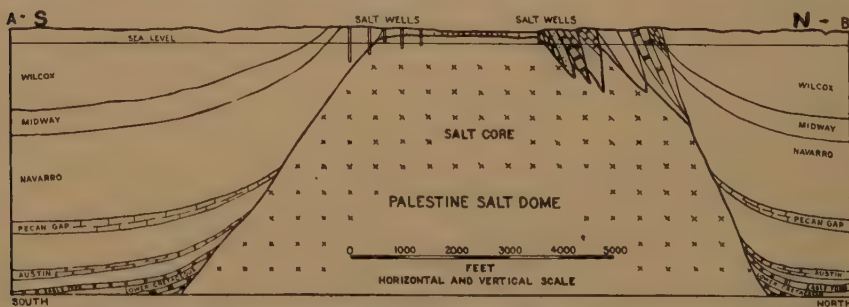


FIG. 2.—CROSS-SECTION OF PALESTINE SALT DOME FROM SOUTH TO NORTH. (After Powers.)

are more nearly vertical, sometimes overhanging. The Palestine dome is of interest geologically because rocks of Lower Cretaceous age are exposed on the surface, having been lifted over a mile from their original bed by the upward thrust of the salt core.

³ S. Powers: Reference of footnote 1, 1.

PRODUCTION AND CONSUMPTION

In 1933 Louisiana ranked fifth among the states in the production of salt. Texas ranked seventh. Production in the two states and the United States for the years 1929 and 1933 are shown in Table 1.

TABLE 1.—*Production of Salt*
Short Tons

	Year	
	1929	1933
United States.....	8,543,560	7,604,972
Louisiana.....	525,840	532,569
Texas.....	Not available	165,603

It is interesting that the salt miners as a whole have been less affected by the depression than almost any other of the mineral producers. Production from the peak of 1929 fell off only 11 per cent, and in Louisiana there was an actual increase. While 1929 figures for Texas are not available, there being only two producers at that time, there was also an increase in production from 1929 to 1933.

The consumption of salt in the form of brine by the three chemical plants mentioned later, going into production this year and in 1935, will increase the production of the two states by about 450,000 tons per year. By the end of 1935 the combined production of the two states will be of the order of 1,200,000 tons per annum.

KINDS OF SALT

The salt in all the domes developed for commercial grades, especially rock salt, is of remarkable purity for a natural mineral. It is prepared and marketed for table and other food uses without any purification. An analysis of the rock salt from the Jefferson Island dome, Louisiana, by James and Breckler, is as follows: moisture, 0.038 per cent; insoluble in acid, 0.004; calcium carbonate, 0.018; salt by difference, 99.940.

The salt in all domes is not as pure as this. In some there is a considerable amount of anhydrite deep within the dome, making it necessary to thoroughly explore the salt by core drilling before beginning any extensive development for the production of rock salt.

The three general types of salt produced are: (1) rock salt, (2) salt evaporated in vacuum pans and (3) salt evaporated in open pans, or grainer salt. Seven companies produce commercial grades of salt. Of these one produces only open-pan-evaporated or grainer salt, three only

rock salt, two rock salt, vacuum-pan and grainer salt, and one rock salt and vacuum-pan salt. Three companies are making or will make alkalis, soda ash and caustic soda, from brines obtained from domes. This is an entirely new development in this area. The first one, the Southern Alkali Corporation at Corpus Christi, Texas, began operations in October, 1934; the second, the Mathieson Alkali Works, Inc., at Lake Charles, La., started in December, 1934, and the third, The Solvay Process Co., at Baton Rouge, La., probably will begin operating about the middle of 1935.

The rock salt is prepared in two general classes or groups of grades, "coarse," from the coarsest grade of rock salt, about $\frac{3}{4}$ in. to 18 mesh; and "fine," from 18 mesh to the finest marketable size, about 50 mesh. There are no exact recognized standards of size although the same designation of size or grade by different producers represents a very similar product. The grades and corresponding limiting screen size are approximately as follows:

Coarse Grades

No. 4 through $\frac{3}{4}$ in. on $2\frac{1}{2}$ mesh.

No. 3 intermediate between No. 2 and No. 4.

No. 2 through $\frac{5}{8}$ in. on $\frac{3}{8}$ inch.

No. 1 through $2\frac{1}{2}$ mesh on 4 mesh.

"A" grade through 4 mesh on 6 mesh.

"C" grade through 6 mesh on 12 mesh.

"Granulated" through 10 mesh on 18 mesh.

Another size sometimes made from the first screening of the rock salt, called "stock," is through 12 mesh on 30 mesh. The "granulated" is usually a reground, kiln-dried product.

Fine Grades

Through 18 mesh on 24 mesh.

Through 18 mesh on 35 mesh.

Table, through 24 mesh on 35 mesh.

The "granulated" grade here placed in the coarse grades is sometimes grouped with the "fine" grades. Another fine grade is sometimes produced, through 35 on 50 mesh, but is not considered a standard size.

Vacuum-pan salt is made only in the fine sizes, as its principal use is for table salt and food products. It would be impractical and difficult to crystallize the salt in the coarser sizes. One producer grades his evaporated table salt through 26 on 32 mesh. A finer grade, through 32 on 50 mesh, is called salt flour, and is largely used in self-rising flour mixtures. One or two grades coarser than "table" are used for various food preparations.

Open-pan or "grainer" salt is made to supply certain customers and uses. It is not prepared in table size. The texture is quite different from rock and vacuum-pan salt. These two have almost cubical grains while grainer salt is flaky, with thin, flat grains. Some grainer salt is

kiln-dried and screened to definite sizes but much of it is shipped unsized after seasoning, draining and air-drying.

In addition to the ordinary grades of granular salt, most of the operators produce special forms and mixtures, one of the most important being salt blocks for stock salting. These blocks weigh 50 lb.; they are formed in a hydraulic press under a pressure of about 20,000 lb. per sq. in. Fine grades of both rock and evaporated salt are used, and must be perfectly dry. Imperfect blocks are sawed into smaller blocks or bricks. Blocks are made plain or sulfurized with a small amount of ground sulfur. Some of the companies produce a mixture known as "smoke salt," for meat curing at home, which contains sugar, seasoning and a pyroligneous acid, and probably niter.

Table grades for household use are packed and shipped in the familiar small pasteboard cartons of rectangular and circular section, filled by automatic machines. A small amount of magnesium carbonate is added to the table grades to prevent caking in the package. Iodized salt is prepared by the addition of a small amount of potassium iodide.

Other commercial grades are shipped in cotton sacks, in amounts from 5 to 200 lb. The sacks are filled and sewed by automatic weighing and bagging machines. Formerly salt was sometimes packed in wooden barrels, but this has almost completely ceased. The coarser grades of rock salt are sometimes shipped in bulk to large industrial consumers.

PRODUCERS

The operating companies in Louisiana and Texas are listed in Table 2, with the names of the domes from which the salt and brine are secured, reported depth to the top of the salt plug and method of extraction.

TABLE 2.—*Domes and Companies*

Map No.	Operating Company	Name of Dome	Depth to Salt, Ft.	Method of Mining
1	Southern Alkali Corporation	Palangana	850	Brine wells
2	United Salt Corporation	Hockley	1010	Shaft
3	Morton Salt Co.	Grand Saline	213	Shaft and wells
4	Mathieson Alkali Works	Hackberry		Brine wells
5	Carey Salt Co.	Winnfield	440	Shaft
6	Palestine Salt & Coal Co.	Palestine	165	Wells
7	Jefferson Island Salt Mining Co.	Jefferson Is.	100	Shaft
8	Avery Rock Salt Mining Co.	Avery Island	20	Shaft
9	Myles Salt Co.	Weeks Island	100	Shaft
10	Solvay Process Co.	Choctaw	667	Brine wells

The Southern Alkali Corporation, the Mathieson Alkali Works and the Solvay Process Co. do not produce salt; they use the brine for the

production of alkalis only. The plant of the Southern Alkali Corporation is on the harbor at Corpus Christi, Texas. It brings the brine from the Palangana dome through a 14-in. cast-iron pipe line 60 miles long; the brine flows by gravity and no pumping is required.

The Mathieson Alkali Works is at Lake Charles, La. Brine is brought to the plant through a pipe line about 20 miles long, from the Hackberry dome.

A plant for the Solvay Process Co. is under construction at Baton Rouge, La. The source of brine is the Choctaw dome, about 14 miles away. It will be piped under the Mississippi River.

The alkali plants all produce their brine from wells at their respective brine fields. The tops of the salt cores of the domes used by the alkali plants as a source of brine are at relatively greater depths than are those mined by brine wells by the salt-producing operators, therefore a different method of well operation is used. The general procedure is to drill the well to the top of the salt, where the casing is set and packed. The well is then deepened 600 ft. or more into the salt and a fresh-water pipe lowered inside the well nearly to the bottom. Fresh water pumped down the water pipe dissolves the salt and because of the pressure on the fresh-water pipe at the surface the brine rises to the surface through the casing. The general arrangement is shown in Fig. 7. The pressure on the fresh-water supply at the surface is that necessary to overcome frictional resistance and the difference between the unit pressures of columns of fresh water and brine at the bottom of the fresh-water pipe.

The United Salt Corporation, near Hockley, Texas, mines salt through a shaft 1525 ft. deep, on the Hockley dome. It produces only the coarse grades of rock salt and stock salt. The depth to the top of the cap rock is 117 ft. The shaft is 22 ft. inside diameter to 135 ft., and to this depth is lined with a concrete drop shaft. From this point to the bottom the shaft is square, 15 by 15 ft., and is unlined. It is divided into four compartments, two for hoisting, one for ventilation and ladderway and one for hoisting men and materials. The general method of mining is the same as that described in detail later for other mines.

The Morton Salt Co., at Grand Saline, Texas, operates both a shaft and wells, and produces all grades of rock and evaporated salt. Details are given later in the paper.

The Carey Salt Co. has a plant on the Winnfield dome, a few miles from the town of Winnfield, La. All grades of rock salt are made. Mining is through a circular concrete-lined shaft 811 ft. deep to the working level. The depth to the salt is 440 ft. Hoisting is done in balanced skips with a cylindroconical electrically driven hoist. A modified room-and-pillar or checkerboard system of mining is employed. This will be described later. The surface plant is one of the most recently built and best constructed.

The Palestine Salt & Coal Co. is on the Palestine dome about 7 miles west of the town of Palestine, Texas. Details of operation will be given later in this paper.

The Jefferson Island Salt Mining Co. is on the Jefferson Island dome, 9 miles west of New Iberia, La. This elevated area, forming the so-called "Island," was formerly the winter home of Joseph Jefferson, the actor, for whom it is named. In 1894 Mr. Jefferson had a well drilled for water, which struck salt at 334 ft. Messrs. Jones and Bayliss acquired the property and in July, 1919, drilled 30 holes for salt and organized the present company. After losing the first shaft, the company completed the second one in February, 1922. Details are given later in the paper.

The Avery Rock Salt Mining Co. is on Avery Island, about 10 miles southwest of New Iberia, La. It is surrounded by marshes and is reached by dirt causeways and a railroad. There is evidence that the existence of brine springs was known and utilized by the aboriginal inhabitants before the coming of white men. The springs were rediscovered and an attempt made to recover the salt as early as 1791, but operations ceased in the early part of the nineteenth century. Following the closing of the first mining operations in 1870, the mine was reopened in 1879, and until 1896 various companies worked the property through other shafts. They were too near the top of the salt, however, and had difficulty with water and caving.

In 1898 the Avery Rock Salt Mining Co. was organized. After test borings a shaft was sunk, which entered the salt at a depth of 54 ft., a few feet above sea level. It was continued to a depth of 518 ft. A second, an air shaft, was sunk in 1922. The main shaft is 21 by 10 ft. and divided into three compartments, two for hoisting and one for service lines of air, electric wires, ventilation, etc. Hoisting is done in balance in mine cars on cages, which are hoisted to the top floor of the mill and dumped. A modified room-and-pillar method of mining is employed. Rooms are 40 to 60 ft. wide by 60 to 80 ft. high. Pillars are 40 to 60 ft. thick. The face at the floor is undercut with short-wall machines. Drilling is done with rotary electric drills. The salt is loaded by electric shovels and hauled to the shaft by electric locomotives. In the mill all grades of rock salt are made by crushing and screening. Evaporated salt is made by dissolving undesirable portions of the rock salt and that too fine for standard grades and crystallizing in multiple-effect vacuum pans.

The Myles Salt Co. is on Weeks Island, 15 miles south of New Iberia, La. Weeks Island is entirely surrounded by marshes and bayous and is made accessible by a dirt causeway. The island rises to a height of about 135 ft. above sea level and has an uneven surface due mainly to solution of the underlying salt followed by slumping. The first serious attempt to develop it was in 1898; by the present operators. The first 100 ft., to the salt, is lined with sectional cast-iron tubular casing. Because of

serious water difficulties, nearly three years was required to complete the shaft. Vaughan⁶ gives the following description of the shaft:

After lowering a cast-iron joint casing 10 ft. into the salt mass, the shaft was continued downwards for 35 ft., the cylindrical form being maintained. The shaft is lined with wood lagging. Surrounding this there is a wall of concrete 1 ft. thick. Then at fairly even intervals there are four rings of asphalt from 3 to 5 ft. in height and with bases 2 to 3 ft. thick. Moreover, the surface of the salt was heated by hand

torches and painted with asphalt. Below the tubular portion, the shaft is square and continues to a total depth of 645 feet.



FIG. 3.—MINE MAP, MYLES SALT CO., LTD., WEEKS ISLAND, LOUISIANA. (From F. E. Vaughan, work cited.)

Mining is by a modified room-and-pillar method similar to the other mines. Rooms are 60 ft. wide, with pillars 60 to 80 ft. thick and crosscuts 150 ft. apart. Power shovels are used for loading and electric locomotives for haulage. At the shaft bottom the salt is crushed into the skip bin and hoisted in a single 5-ton skip. A map of the mine (1925) illustrating the arrangement of the rooms or stopes is shown in Fig. 3. All grades of rock salt are prepared in the surface plant by the usual methods of crushing and screening.

The operations of three companies are described with more detail, as being typical of operations producing only rock salt, combined rock salt and salt from brine wells and from wells alone.

JEFFERSON ISLAND SALT MINING CO., JEFFERSON ISLAND, LOUISIANA

At Jefferson Island the circular shaft, 25 ft. in diameter and lined with reinforced concrete, is 900 ft. deep, 800 ft. to the working level, and enters the salt 100 ft. below the surface at a point 40 ft. below sea level. All grades of rock and evaporated salt are produced.

The shaft was deepened 100 ft. below the working level to allow the skips to descend to a position for loading from the underground storage bin cut in the salt below the working level. Salt is hauled from the loading places in 5-ton side-dump cars to the shaft by electric trolley locomotives equipped with an extension cable for operation beyond the

⁶ F. E. Vaughan: The Five Islands of Louisiana. Amer. Assn. Petr. Geol. (1925) 9, No. 4, 779.

trolley wire. At the shaft the cars are dumped directly into the hopper of a roll crusher, which reduces the salt to about 6 in. From the crusher the salt drops into the 1600-ton bin from which the skips are loaded. These hold 6 tons and operate in balance. The skips are hoisted by an electric hoist with a cylindroconical drum.

The method of mining employed is a type of room and pillar, or large open stopes similar in plan to that shown in Fig. 3. Rooms or stopes are driven 60 to 80 ft. wide and carried up to a height of 70 to 90 ft. Pillars between the rooms are 60 ft. thick, with crosscuts as needed of the same width as the rooms. When a room or crosscut is started, the face is is

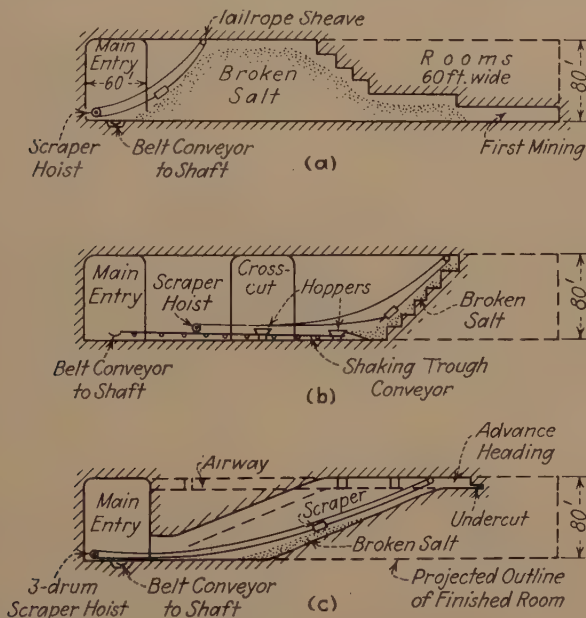


FIG. 4.—ROOM MINING METHODS, JEFFERSON ISLAND SALT MINING CO., JEFFERSON ISLAND, LOUISIANA.

first undercut the full width at the floor by a short-wall chain undercutter similar to those used in coal mining, electrically driven. The depth of the undercut is 6 ft. and the cross travel of the cutter about 6 in. per minute. The face is then drilled and blasted down for a height of 10 ft. This height or slice is carried through to the next opening or room, and the salt loaded by hand into mine cars. The roof is drilled and blasted down on to the floor in slices 15 to 20 ft. high, the miners working on the broken salt to reach the roof or back. When one slice is completed another is started, and so on to the desired height. All slices cannot be carried through to completion, as there is not room for the broken salt, so the entrance to the room is carried up first to the full height and the salt loaded out from the toe of the slope of the broken salt as the room fills.

Fig. 4a illustrates the method of mining, except that the salt is loaded with a shovel instead of being removed with a drag scraper as shown in the cut. Drilling is done with electric rotary drills mounted on columns. The salt is loaded into the mine cars by an electric shovel with a $\frac{3}{4}$ -yd. dipper that stands on the floor of the mine. This is a full revolving shovel mounted on caterpillar treads, as shown in Fig. 5.



FIG. 5.—LOADING ROCK SALT WITH $\frac{3}{4}$ -YARD ELECTRIC SHOVEL, JEFFERSON ISLAND SALT MINING CO.

No timbering is required in any part of the mine. There is no water to be pumped, and dust conditions are not bad, except at times near the face when the undercutter is working. The small amount of dust made is not harmful.

At the surface the skips dump into a small pocket that feeds a roll crusher reducing the salt to about $\frac{3}{4}$ in. The salt is then elevated to the top floor, from which it descends over the screening system and is graded into all commercial sizes. Hum-mer screens are used for sizing and elevators and conveyors are arranged so that various sizes can be bypassed or sent to the fine-crushing rolls for further reduction, to suit the variation in market demands.

In preparation of the fine sizes, the salt is dried in a gas-fired, indirect rotary kiln before final screening to the finished grade. From the screen the salt passes to bins feeding automatic weighing and packaging machines on the second floor, from which the finished packages are chuted to the loading floor below.

To meet certain market specifications and demands evaporated salt, both vacuum-pan and grainer, is prepared. The brine is made from the salt that is too fine to market, other surplus salt, or when necessary finely

crushed rock salt, dissolved in a series of troughs in hot water. The brine is thoroughly agitated and pumped to a settling tank, a basin on the highest point of land near the plant.

The clarified brine flows to double-effect evaporators for crystallization to vacuum-pan salt or to three open-pan evaporators for grainer salt. The vacuum-pan salt is dewatered on an Oliver continuous filter and passes by conveyor belt to bins feeding a rotary indirect-heat drier or to storage piles. The dried salt, after passing through a cooler, goes to final sizing screens and to packing bins.

The three open-pan evaporators are each about 14 ft. wide by 130 ft. long by 30 in. deep. Steam pipes submerged in the brine evaporate the water at atmospheric pressure, the salt crystallizing out, partly on the surface in flakelike crystals until they are heavy enough to sink to the bottom. A system of automatic reciprocating rakes pushes the salt along the bottom to the discharge end and up to an inclined drain board. The next forward movement of the rakes pushes the salt on to a conveyor belt passing along the discharge end of all the pans. In the pans the steam pipes are hung by supports from above and the rakes work underneath them. The belt conveyor deposits the wet salt in large piles in the storage room for draining and seasoning. From here it is recovered as needed, put through the rotary kiln for drying and graded on nearly horizontal Rotex screens of the shaking or oscillating type. These screens are less likely to break up and destroy the flaky texture that is desired than are inclined vibrating screens.

The company has its own boiler plant fired with natural gas, to furnish steam for the evaporators and power for the operation of the mine and mill. Salt shipments are made over a branch of the Missouri Pacific lines serving the plant from New Iberia.

MORTON SALT CO., GRAND SALINE, TEXAS

The Morton Salt Co. operates a shaft mine, known as the Kleer salt mine, on the southeastern part of Grand Saline dome, and brine wells and an evaporating plant on the northwestern part of the dome, in the town of Grand Saline. The two plants are about a mile apart. The wells are about 265 ft. deep to the salt and penetrate the salt about 200 feet.

Salt has been extracted from Grand Saline dome for a long time and by different companies. The Morton Salt Co. is the only one now operating. Its brine plant has been in operation many years; the shaft plant for the production of rock salt was completed in 1931. Many difficulties have to be overcome in sinking the shaft through the heavy water-bearing strata overlying the top of the salt⁴.

⁴ Details of sinking this shaft are given by M. Taylor [*Min. & Met.* (Dec., 1930)] and shaft equipment and hoisting are described by H. B. Cooley [*Eng. & Min. Jnl* (May, 1932) 256].

Mine Plant

The shaft enters the salt at a depth of 213 ft. The total depth to the working level is 700 ft. The shaft is deepened below this to accommodate the skip below the loading chute from the working level. The circular shaft is 14 ft. 6 in. in diameter inside the reinforced concrete lining, which extends to the working level. There are two hoisting compartments, one for the skip and one for the counterbalancing cage.

One segment of the circle outside the rectangle formed by the hoisting compartments is partitioned off with an airtight partition and serves

as the upcast for ventilation. Salt is hoisted in a skip of 4 tons capacity. The skip and its load are partly counterbalanced by the weight of the cage. Hoisting is done with a double-reduction geared electric hoist with a double cylindroconical drum, of which the small diameter is 7 ft. and the large diameter 9 ft. The maximum hoisting speed is 580 ft. per minute. By an ingenious arrangement of the lengths of the smaller cylindrical portions the skip travels from its loading position in the sump to its dumping position in the headframe while the cage travels from the surface landing to the working level. This is accomplished by having a greater number of turns of the cage rope



FIG. 6.—REINFORCED CONCRETE HEADFRAME. MORTON SALT CO., KLEER MINE, GRAND SALINE, TEXAS.

winding on the smaller cylindrical diameter. The large diameter is common to both ropes. By this method cage service from the surface to the working level can be given with no time or trips lost in hoisting salt.

The headframe (Fig. 6) is of reinforced concrete with the panels of the vertical part of the frame closed in with concrete walls. The concrete covering of the structural shapes of the headframe not only adds strength but protects the steel members from the corrosive effect of the salt. The structure housing the receiving hopper for the skips and secondary crusher and picking tables is an integral part of the headframe. The weight of the skip and contents is 16,000 lb. The rope diameter is $1\frac{1}{4}$ inches.

Primary crushing to 6 in. is done at the shaft bottom by a double roll crusher, which discharges to the pockets loading the skips. There is no underground storage at the shaft bottom.

The plan of the mine openings does not differ greatly from other mines in the salt domes. A room, or main entry, was started in opposite directions paralleling the shaft, but has been developed in only one direction. After leaving the shaft bottom this opening was enlarged to the full size of the rooms or stopes, 60 ft. wide by 80 ft. high. The method of enlarging and advancing was that shown in Fig. 4c. An inclined heading is driven at a slope of 20° from the horizontal and is carried up to a vertical height of 80 ft. and the full width of the room. Starting at the toe of the slope, the floor is then drilled and blasted and the salt pulled down by a bottomless tail-rope scraper of 2 cu. yd. capacity, operated by a three-drum electric hoist. At the beginning of operations this scraper delivered directly into the hopper over the roll crusher at the shaft bottom but when the distance from the shaft increased a 36-in. belt conveyor was installed, which delivers to the crusher hopper and receives the salt brought to it by the scraper near the face. With the three-drum scraper hoist, two tail ropes can be employed, one leading from each corner of the face, so that the scraper can work over the entire width of the room. After the main entry had advanced far enough, rooms were turned at right angles. The original plan was to open these with a narrow neck, but now they are started the full width. An inclined heading was driven up to the desired height, then advanced horizontally, and mining started on the floor of the slope. The belt conveyor was extended, the hoist moved to a point opposite the room and the drag scraper delivered to the belt as shown in Fig. 4c.

Ventilation is obtained by a small heading starting from the upcast compartment of the shaft, paralleling the main entry and at the level of the top of this. Laterals then parallel each room.

Another method was then tried, as shown in Fig. 4b. The room is started full size, 60 ft. wide by 80 ft. high, and advanced by a series of benches with the angle of the advance of the series equal to the angle of repose of the broken salt. The salt was pulled down the slope and to the conveyor with the drag scraper. This method is not entirely satisfactory because it is necessary to muck off the top of each bench by hand before it can be drilled.

A third method, now used, is similar to the one described for the Jefferson Island mine (Fig. 4a). The heading or breast at the floor level is driven 16 ft. high and carried forward the full distance, and the salt is all removed. The roof is then taken down in a series of slices, each 12-ft. high advance by two cuts of 6 ft. each to facilitate drilling. The salt remains in the room until the top is reached, the drillers working on the broken salt.

In all methods crosscuts are driven between the rooms at regular intervals. In methods *a* and *b* these are usually as wide as the room and the method of driving them is the same.

By using angle sheaves at points opposite the crosscuts, the same hoist is employed to operate the scraper in them as well as in the main room. This is made possible by the use of a shaking trough conveyor in the room, which delivers to the main belt conveyor to the shaft. The scraper delivers to hoppers over the trough from both the crosscuts and room face. No cars are used in the mine.

Undercutting of the salt in the advance headings is done with a short-wall undercutter with a 7-ft. bar, which undercuts to a depth of 6 ft. The advance across the face is 6 in. per minute. Drilling is done with rotary electric drills. Blast holes are placed 4 ft. apart horizontally and 3 ft. vertically in the face. They are drilled $6\frac{1}{2}$ to 7 ft. deep. From $\frac{1}{2}$ to $\frac{3}{4}$ lb. of 35 per cent dynamite is used per ton of salt.

Surface Plant

The mine skip dumps into a small bin at the side of the headframe from which a reciprocating pan feeder delivers the salt to a grizzly with bars spaced about 1 in. apart. The oversize passes over a shaking picking table where discolored salt, rubbish, etc., is picked off. The coarse salt is delivered to a 24 by 30-in. double roll crusher and joins the bypassed fines through the grizzly. The combined product then is delivered by an 18-in. inclined belt conveyor to the mill, where a bucket elevator raises it to the top of the first set of screens. Hum-mer vibrating screens are used throughout.

The oversize from the first screen or any of the other coarser sizes not desired passes to a rotary recrusher and is returned to the main elevator. The natural mine fines, finer than 50 mesh, are next screened out and pass to a bin feeding the brine solution tank or for bulk loading. The mixed granular grades are then elevated to the second series of screens. These make the various grades of coarse salt down to 12 mesh. The salt finer than this goes to the fines bin but sometimes may be used to make a special grade for stock use, sized between 12 and 30 mesh.

The various grades of "coarse" salt pass by chutes or belt conveyors to their respective storage bins. The required amount of coarse salt goes to roller mills for regrinding and then to the second set of screens making the finer of the "coarse" grades and a product containing the mixed "fine" grades. The latter product is sent to a bin feeding a gas-fired indirect-heat rotary drier. The dried salt then goes to the third set of screens making the "fine" grades. These are sent to their respective bins. The necessary amount of table grade required for treating passes through a mixer, where the required amount of iodide is added.

The dust and fines from the fine screening plant go to a dust bin and with the fines from the first screening are put into brine form and piped to the evaporating plant one mile distant, or are pressed into salt blocks.

The packing machines for cartons and bags are under the storage bins and so arranged that they can be fed by the different grades as desired.

Evaporating Plant

The evaporating plant receives brine from the rock-salt plant and from wells near by. Before the mine was operated, the entire supply was drawn from wells.

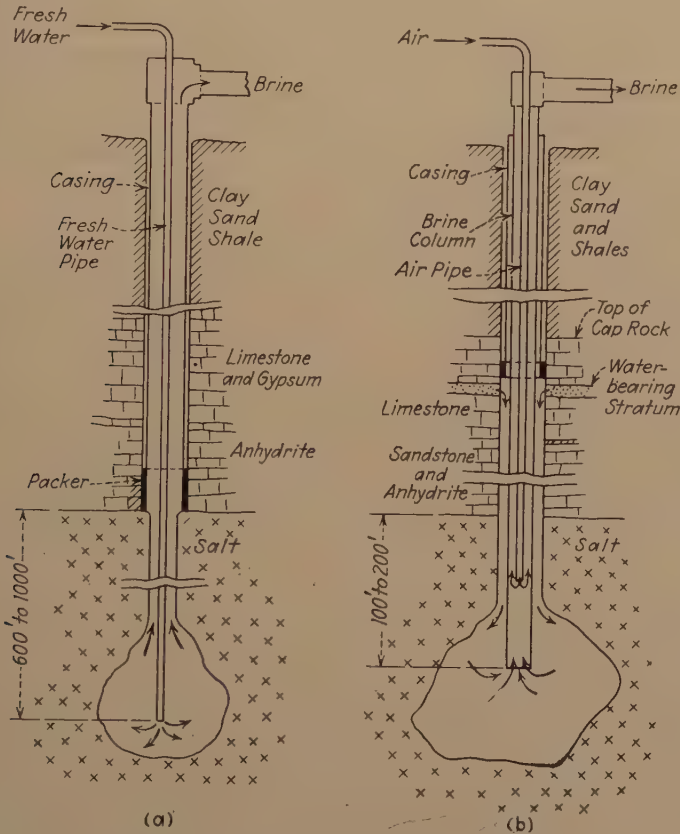


FIG. 7.—GENERALIZED ARRANGEMENTS OF BRINE WELLS.
a. Hydraulically operated. b. Operated by air-lift.

The wells are 400 to 500 ft. deep. In drilling, the well is finished into the top of the cap rock, usually limestone, and a casing set and packed or well sealed into the cap rock. The well is then drilled on into the salt for a depth of 100 to 200 ft. below the top of the salt. The cap rock contains a stratum, usually called a "cavity," which is strongly water-bearing, and the casing is set above this. A 4-in. brine line is then placed inside the casing nearly to the bottom of the well in the salt. Inside this a 1 $\frac{1}{4}$ -in. air pipe is set, its lower end being placed at a depth depending on

the amount of lift required or the height to which the brine will naturally rise in the well. The ground water from the "cavity" flows down the well outside the brine column, dissolves the salt up to the saturation point, about 25 per cent, and is raised to the surface through the brine column by the air-lift effect of the compressed air released from the air line. An air pressure of 100 lb. per sq. in. is used. At Grand Saline the depth to the "cavity" is about 220 ft. and to the top of the salt about 265 ft. The intervening depth is mostly anhydrite. A generalized arrangement of a brine well operated by air-lift is shown in Fig. 7.

At the surface the brine is pumped into storage tanks, where it is allowed to settle. It is then treated to remove impurities, mostly calcium sulfate. The refined brine is then evaporated in triple-effect vacuum pans or open grainer pans. There are two sets of vacuum pans and two open pans. The grainer pans are 15 ft. wide by 90 ft. long and each has a capacity of about 7 tons of salt per day.

The salt from the vacuum pans is dewatered in centrifugal separators and then goes to a gas-fired indirect-heat rotary drier followed by a cooler. It is graded to commercial "fine" sizes on vibrating screens and stored in bins above the bagging and carton-packing machines. The grainer salt goes to storage piles on the storage floor, from which it is bagged or barreled or sometimes kiln-dried before shipping. Salt too fine for standard grades is made into salt blocks, either plain or sulfurized, or marketed as flour salt.

Steam for the evaporators and for power is produced in boilers fired by lignite, which is mined 9 miles from the plant and brought in on a branch railroad.

PALESTINE SALT AND COAL CO., PALESTINE, TEXAS

The Palestine Salt & Coal Co. began operating a brine-evaporating plant for the production of salt on the Palestine dome in 1904. The brine is obtained from wells. The wells are drilled into the top of the cap rock at a depth of 140 to 200 ft., the casing is set and packed or sealed, then the wells are deepened 200 to 250 ft. into the salt. A brine pipe and air pipe are lowered into the well as described for the Morton Salt Co. at Grand Saline and as shown in Fig. 7. A water-bearing sand under the cap rock supplies the inflow of water. The air-lift raises the brine to small elevated tanks near the wells, mounted high enough so that the brine flows by gravity about $\frac{1}{2}$ mile to the plant, across a small, shallow lake.

At the plant the brine is first treated with soda ash and lime and allowed to settle in large wooden storage tanks. It then is filtered through a coarse gravity filter and stored in the purified brine storage tanks, also of wooden construction.

As needed, the brine is drawn off into preheater tanks in which the brine is heated by exhaust steam from the heater coils of the grainer pans. From the preheater tanks the brine flows to the grainer pans. There are seven of these, each 14 ft. wide by 140 ft. long by about 30 in. deep. The method of operation has been described under the Jefferson Island plant. The capacity of the grainers is rated at 15 tons each per day.

The salt from the pans is sent to the drainage floor for storage. As needed it is transported in hand buggies to a steam-heated rotary drier, cooled, screened and stored in the dried-salt bins ready for packaging in barrels or bags. Small cartons of table salt are not packed. Salt blocks, both plain and sulfurized, are made from the fine dried salt.

Lignite is used for the steam-generating plant, which furnished steam for the grainer pans, the steam drier, compressor and for power generation. The company has its own lignite mine about two miles from the salt plant.

DISCUSSION

(*S. H. Dolbear presiding*)

MEMBER.—What per cent dynamite is used in blasting salt?

W. M. WEIGEL.—About 35 per cent.

D. C. BARTON,* Houston, Tex. (written discussion).—Perhaps the oldest, an interesting, but minor salt industry in Texas is not mentioned by Weigel. The Sal del Rey saline in Hidalgo County, Texas, was an important source of salt in the early days of the settlement of Texas and of northeastern Mexico. Römer's map of Texas of the date 1847 shows roads converging on Sal del Rey from the Mexican cities of Mier and Reynosa and from the north.

Sal del Rey is the second largest of the many wind-scooped lakes and ponds in the belt of Lissie in southern Texas. The upper ground water of that area is distinctly saline. The basin of Sal del Rey extends below the normal water level of those waters; and under the hot semi-arid climate of the area, almost continuous concentration of brine takes place. The lake dries up completely during the summer and deposits a layer of salt several inches thick. Several years ago, my assistant and I found the section through the salt to be a layer of salt 2 in. thick at the surface; under it 4 in. of black clay, then 3 in. of clear rock salt and under it 2+ in. of black clay again. But according to the accounts of local Mexicans, there is 16 ft. of solid salt in the center of the lake. The diameter of the lake is approximately one mile. Small wagon trains still come in from Mexico; the Mexicans break up the surficial layer of salt, load the cakes on their wagons, and take the salt back into the interior of northeastern Mexico, presumably just as they have done for the past century and more. The tonnage of salt won in this manner is presumably small.

In connection with the numerous salines and salt springs at which salt was evaporated before and during the Civil War, in Louisiana and Texas, the statement is made that the location of these in nearly every instance was near what now is known to be a salt dome, and that the source of the salt in the brine was in the salt core. In the search for oil in southern Louisiana and southeast Texas, I have investigated a number of places of which the natives reported the evaporation of salt during the Civil War.

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One of the localities was on what now is known to be the Tomball salt dome, but the depth to the salt probably is 9000 or 10,000 ft.; the faint brine seepages may be of deep-seated waters coming up faults (the Hockley fault?) or may be produced by evaporation of normal ground water brought to the surface by capillary action in the Lissie sands. The other localities that I investigated are known not to be associated with shallow salt domes (depth to the salt less than 5000 ft.) and have not yet been shown to be associated with deep salt domes. There were, of course, brine springs on such shallow domes as Avery Island and Sour Lake.

At Anse la Butte, St. Martin Parish, Louisiana, salt was produced by the brine method from 1922 to 1930 by the Lafayette Salt Co. and the Star Salt Co. The latter company piped the brine from the Anse la Butte dome to a plant near the Baldwin Lumber Company's sawmill near Lafayette and utilized waste heat from the sawmill for evaporation. The total production of salt from Anse la Butte is given by Howe and Moresi as 120,000 short tons.

The salt reserves of Texas and Louisiana are stupendous. Several years ago, in connection with the economic importance of salt domes,⁵ I calculated the reserves of the Texas-Louisiana salt domes, without regard to the enormous Permian salt deposits of West Texas, to be 10 cu. km. at depths shallower than 1000 ft., 40 cu. km. at depths shallower than 2500 ft., and 100 to 150 cu. km. at depths less than 5000 ft. The easily minable reserve of the Texas-Louisiana salt domes would suffice to supply the present world consumption of salt for 1000 years; and such mines as those at Weeks, Avery, and Jefferson Island have in sight easily minable reserves for 25,000 years at the present rate of production.

⁵ University of Texas *Bull.* 2801 (1928) 7-53.

Some Recent Developments in the Use of Sodium Chloride (Common Salt)

By C. D. LOOKER*

(New York Meeting, February, 1936)

COMMON salt is mentioned in the most ancient writings as an important article of diet. It is fairly certain that it was used by men and animals long before the dawn of civilization. Its presence in natural springs has often been the determining factor in colonization; wars have been fought for possession of salt deposits; trade routes have been established expressly for the purpose of traffic in salt. Naturally this led men into adventures and discoveries that altered history and advanced civilization.

In the early days the uses for salt were few and the methods of manufacture were crude and the purity low when compared to salt of the present day. Then the salt came largely from springs, shallow wells or the ocean, while today deep wells must be drilled to the natural brines or beds of rock salt and spacious shafts sunk so that the ancient deposits may be made to yield their supply of stored mineral.

MINING AND PURIFICATION

The quantity of available salt seems inexhaustible. It occurs in practically every country of the globe. About 20 million tons is produced and consumed annually in the world. Of this amount about 7 million tons is produced in the United States; $1\frac{3}{4}$ million tons of rock salt, $2\frac{1}{4}$ million tons as evaporated salt and $3\frac{1}{2}$ million tons in brine used in the chemical industry.

Mined rock salt is used in its natural state without purification. It is classified for the trade by crushing and screening. The average purity of rock salt mined in the United States is about 98.3 per cent pure sodium chloride¹. The usual impurities are gypsiferous shale, anhydrite and silica, with traces of potassium, magnesium and calcium compounds.

Refined or evaporated salt is made by crystallizing the salt from brine in open pans or vacuum pans. A comparatively small amount of salt is made from ocean water or the brines of inland lakes or seas, sometimes by solar evaporation. By far the larger amount of brine is obtained by drill-

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¹ References are at the end of the paper.

ing wells to the beds of rock salt and dissolving the salt with water pumped down from the surface. (This might be called hydraulic mining.) The saturated brine is pumped to the surface, purified, settled and subjected to evaporation.

Salt crystallizes in the cubical system. Vacuum pans produce individual cubes; ordinary table salt is a good example of vacuum-pan salt. Salt produced in the open pan forms in the surface of a hot brine and consists of so-called flakes, each one built around a cubical nucleus forming a hopper. The sides of the hoppers break along straight lines and are composed of thin flakes made up of cubes.

USES OF SALT

Salt in some way or another, either directly or indirectly, enters into the manufacture or preparation of a majority of the common commodities we encounter in life's daily routine. To enumerate all the uses of salt or give a complete account of the influence of salt on industry lies beyond the scope of this paper. It is my purpose merely to mention briefly some of the more recent developments and possibilities.

Salt was the source of the chlorine that bleached the cotton in the bed covers and the pulp in the morning paper. The electrolytic companion of chlorine is caustic soda or lye, which helped salt itself to make the soap for the morning bath, the lather for shaving, and the detergent for the laundry. The clothes we had dry-cleaned with a chlorinated solvent were dyed in all probability with the aid of salt to exhaust the dye bath.

The drink of water we took this morning may have been softened by zeolite regenerated with salt brine and made germ-free by the addition of chlorine, or it might have been softened by soda ash made from salt.

The lacquer on the bodies of our automobiles, the glass in the windows and windshield and the rolled iron in body and frame had salt or compounds made from it used in their fabrication. The grease and oil in the bearings and crankcase, in all probability, were extracted with chlorinated hydrocarbon solvents. The bearings themselves may have been casehardened in molten salt, salt mixtures or preparations consisting of sodium compounds. In ethyl gasoline, the bromine for ethyl bromide was probably liberated from sea water by the use of chlorine. This is truly returning part of the salt of the earth to the sea from whence it came. Leather is preserved and tanned by the use of salt. In time we may ride on tires made of synthetic rubber, of which the raw materials are salt, coal and limestone.

Paper Bleaching and Extraction of Oils.—Extra demands are being made for chlorine at the present time to bleach sulfate paper pulp and make chlorinated hydrocarbon solvents for the extraction of lubricating oils from petroleum. This has stimulated research to find new methods of making chlorine from salt without overproducing caustic soda. One

method² of prominence that has passed the pilot-plant stage is the interaction of salt and nitric acid obtained by the oxidation of synthetic ammonia. This process makes chlorine and fertilizer grade of nitrate of soda. The increased production of paper by the sulfate process opens an extended market for "salt cake," sodium sulfate, a limited amount of which is manufactured in the United States from rock salt and sulfuric acid. The excess demand is supplied by importations from Germany, Belgium, Canada, Chile and The Netherlands. The limitation on the manufacture of salt cake is the demand for hydrochloric acid. Interesting possibilities are being studied to utilize this byproduct.

Curing Hides.—In South America new technique has been developed in curing heavy hides³. Before slaughter the animals are made to pass through a spray of water. After the hides are removed they are scrubbed and soaked in salt brine before curing in salt packs completely surrounded by salt held in place by a wooden frame. This has replaced the usual open pack and produces hides of superior quality that compete in the world market with hides cured by the older methods. Because of economic reasons, the South American method has not been universally adopted in this country. Some of our packers cure by soaking in brine hides that are to be tanned immediately.

Diet.—It is estimated that the per capita consumption of salt in the United States in 1935 is about 65 lb. annually, but of this amount only about 29 lb. is used in food and in its preparation, the remainder being used in industry. A recent survey indicated that each person adds about 6 lb. of salt annually as a condiment to his food and there is present an additional 6 lb. as a natural ingredient or added for preservation. Salt is considered to be an essential constituent in the diet of animals. It is said that few experiences are more annoying than "salt hunger"; the Chinese hunger torture consists in giving prisoners ample food in the form of rice without salt.

Salt is added to food mainly (1) to satisfy an acquired taste, (2) to preserve food and (3) to maintain the salt equilibrium in the tissues and blood stream. Carnivorous animals or the hunter who lives mainly on the meat he kills do not require additional salt, but herbivorous animals do. The amounts required vary with different individuals and different diets, depending upon pathological conditions and the habits of the individual. In general, the amount consumed must be sufficient to replace that lost by excretion, in order to maintain the correct salt balance without calling upon the tissues. It has recently been discovered that the salt balance in the body is controlled by the outer covering of the suprarenal glands, known as the suprarenal cortex⁴. The suprarenal glands lie above and back of the kidneys; the malfunctioning of the cortex results in what is known as Addison's disease. Intravenous injections or larger doses taken inwardly by mouth of a mixture of salt and baking soda or certain organic compounds give lasting relief.

Heat cramps among foundry workers and others exposed to excessive heat are caused by disturbing the salt balance in the blood that bathes the brain. This occurs when the salt lost in perspiration greatly exceeds the reserve supply in the system. Clinical studies and experiments at Boulder Dam and various foundries⁵ show that heat cramps can be greatly reduced in number or entirely eliminated by requiring workmen

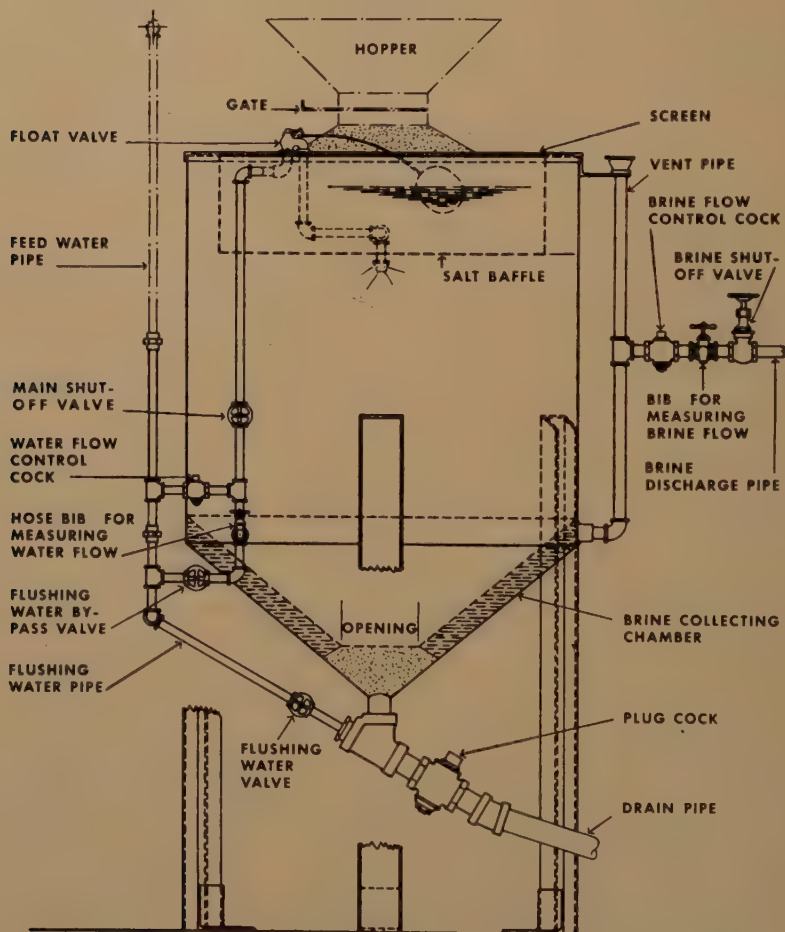


FIG. 1.—DIAGRAM OF LIXATE DISSOLVER.

to take a pinch of salt or a salt tablet with each drink of water when working under extreme heat conditions.

Of particular interest in scientific and medicinal circles at the present time is the making of common salt radioactive by bombardment with deuterons or the nuclei of the atoms of heavy hydrogen.⁶ It is thought that radioactive salt may be beneficial as a substitute for radium because

of its shorter life. This discovery has a large amount of human interest but is insignificant from the standpoint of tonnage for the salt companies because of the small quantity involved.

Dissolution.—For some industries rock or evaporated salt is used as solid undissolved salt—for example, in the production of metallic sodium one process electrolyzes molten salt—but a great amount of salt is used in industry in the form of brine made by dissolving the solid salt in water. In the past this was accomplished mainly by agitation with paddles or compressed air, sometimes by man power; always a slow, tedious process producing a cloudy brine, especially from rock salt, that had to be filtered for most exacting uses.

The Lixate process has been developed to overcome dissolution and filtration difficulties; it is an automatic method of producing a pure, clear, uniformly saturated brine from rock salt. This brine has been shown to be as free from bacteria as evaporated salt and may be used for all purposes not requiring extreme refinements⁷. The Lixate process combines a method of rapid dissolving without agitation and a method of self-filtration. The entire operation takes place within a dissolver. Water enters through a spray nozzle at the top, percolates through the recommended grade of coarse rock salt, and in its downward course becomes fully saturated brine, which rises within a conical chamber and then flows to the storage tank (Fig. 1).

Any insoluble material present in the salt, such as shale, silica or anhydrite, together with any foreign matter that may get into the salt during handling, is automatically removed by self-filtration. The specified grade of rock salt, together with the insoluble contents, acts as a filter.

Brine from rock salt is now being used to salt out dyes and soap, pickle meat, regenerate zeolite, cure pickles and olives, can vegetables, clear oleomargarine, and in various other food processes.

Salt Ice.—At one time a mixture of salt and ice was the only practical means of obtaining low-temperature refrigeration. It is still used on a large scale, especially in cooling refrigerator cars.

A new method has been found to use salt and ice to produce a uniformly low temperature efficiently and economically, by freezing eutectic brine on a FlakIce machine in the form of thin ribbons. Eutectic brine is made by diluting saturated brine to eutectic composition containing 23.3 per cent salt and 76.7 per cent water. Freezing takes place on the outside of thin Monel metal bands, which are a part of a refrigerated drum rotating in eutectic brine (Fig. 2). By means of a change in curvature of the flexible bands, the ribbons of eutectic ice peel from the drum and drop to a refrigerated chamber, to be used as small broken pieces or compressed into a block and used in the same manner as ordinary ice or dry ice.

Eutectic ice, or Salt Ice⁸ as it is called by its producers, is a homogeneous mass of salt and ice crystals that has true eutectic characteristics. It melts at a uniform temperature of -6°F. with the absorption of 101.5 B.t.u. of heat. Salt Ice is handled much more conveniently and costs less than the usual mixture of salt and ice that produces an



FIG. 2.—TOP VIEW OF FLAKICE MACHINE, SHOWING RIBBONS OF SALT ICE LEAVING THE ROTATING CYLINDER.

uneven temperature above zero and has a heat-absorbing capacity of only approximately 83 B.t.u. Three pounds or slightly less of Salt Ice have as much heat-absorbing capacity as one pound of dry ice when used for low-temperature refrigeration. The cost of producing 3 lb. of Salt Ice by the user is about three-fourths of a cent as against 2 to 4¢ per pound of dry ice delivered.

As a part of a series of exhaustive tests conducted by the Salt Ice Corporation, a refrigerator car using Salt Ice was loaded with fish at Gloucester, Mass., by the General Seafoods Corporation, and sent to Detroit. Despite hot weather, the load was maintained at a temperature of approximately 4° F. during the 70-hr. run.

Salt Ice is being produced on a commercial basis by the Salt Ice Corporation of Brooklyn, N. Y.

Stabilized Roads.—The most recent use for salt is in the construction of stabilized roads of the farm to market type. The stabilization of sand and gravel roads with clay or top soil has been in progress since about 1906 but added interest has recently been developed by the Bureau of Public Roads in cooperation with the several state highway departments⁹. Standard specifications have been developed for sizes of materials that produce best results. There is a wide variety in the choice of materials for stabilized roads and considerable latitude in the proportions of different sizes of materials.

The materials of construction most commonly used to secure stability are gravel, crushed stone, slag, sand or stone dust and natural soil binder, which includes fine sand, silt and clay. These materials are either premixed or mixed in place in proper quantities and proportions to give all-weather stability. Clay, silt and sand provide a natural stabilizer for roads when mixed with coarse aggregate, but natural soil does not make a lasting binder. Clay is subject to fairly wide limits of expansion and contraction when alternately dampened and dried, as by rain, wind and sunshine. Alternate expanding and shrinking cause the binding material to break away from the coarse aggregate and make a loose, floating surface. The dry clay and fine sand turn to dust and are blown away by wind or cast aside by traffic. Repeated alternate expanding and shrinking may displace the materials of construction to such an extent that the road will gradually lose its compaction and will have soft spots in wet weather and raveling in dry weather.

Dry clay cannot be compacted to as dense a mass as moist clay. Compacted dry clay soon loses its density because the clay particles are held apart by layers of air. Wet or damp clay lends itself to compaction because the air layers have been replaced by films of moisture, which not only permit closer contact but resist separation of particles as excess moisture is lost by evaporation. Solutions of electrolytes such as common salt aid in removing air films and wetting soil grains, thus assisting in compaction, and soil moistened with sodium chloride solution has a greater retentive power for moisture than soil moistened with plain water¹⁰. The greater compaction seems to be due also to certain electrolytic effects that remove or alter the gel-like films that surround colloidal clay particles, permitting them to unite in closer contact with each other, surrounded by thinner and consequently more tenacious solution films.

The value of the greater retentive power for moisture and of the electrolytic effects appears to be apparent throughout the life of the road but these seem to be most useful during the construction period, when the separate ingredients are reaching equilibrium in the interlocked structure of which the compacted mass is composed. These effects are brought about mainly because the soil binder in the mix is maintained in the plastic condition for a longer period of time during the period of compaction and setting up. This not only adds to the stability of the road but serves to retain the salt over long periods, making frequent additions unnecessary. Properly constructed salt-soil stabilized roads may serve



FIG. 3.—STABILIZED ROAD AT PORTAGEVILLE, N. Y.

for an indefinite time as wearing courses or they may serve as base courses for a higher type of surfacing. Fig. 3 shows a salt-stabilized road at Portageville, N. Y., built in the summer of 1934, using 2 lb. rock salt per square yard of surface 3 in. thick. It was used without maintenance until August, 1935, when it was scarified, salted (10 tons per mile) and reshaped.

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APPENDIX

May 31, 1938

Since this paper was written, in 1936, the developments in the use of salt in stabilized road construction warrant further comment. At that time the work was largely in the experimental stage.

In the few years that roads treated with sodium chloride have been constructed, the mileage has steadily increased each year until there are more than 1000 miles in the United States and a considerable mileage in Canada, including provincial, state, county and township roads, village and city streets and salt-stabilized base courses under bituminous and concrete surfacings. The construction has not been confined to any one locality but has been extended to include at least 20 different states.

Because of practically complete absence of frost boil difficulties in salt-soil-stabilized roads, considerable interest is developing in treating with salt the subsoil beneath the gravel foundation for bituminous surfacings to lower the freezing point of subsoil moisture.

Perhaps the largest mileage of salt roads has been made of pit-run material including soil overburden. In addition to this, other road-building aggregates have been used, such as shale, granular slag and limestone screenings with fine limestone dust, which act as binding material instead of the usual clay mentioned in the paper. The increased solubility of limestone in sodium chloride brine adds to its natural cementing power. Roads made of salt-treated limestone screenings have been called salt-bound-stabilized macadam and are serving both as wearing courses and base courses for higher types of surfacing.

DISCUSSION

[For discussion of this paper, see page 437.]

Use of Sodium Chloride in Road Stabilization

By H. RIES,* MEMBER A.I.M.E.

(New York Meeting, February, 1936)

THE stabilization of a gravel road with soil is a method of treatment developed in recent years by the United States Bureau of Public Roads for the purpose of binding the constituents together better, and maintaining a more durable surface.

The process consists in adding to the gravel or crushed stone and sand, a soil mortar, so-called, of fine sand, silt and clay, mixed in the proper proportions. The coarse gravel and sand supply structural strength and hardness; the fine sand adds embedment support to the coarse sand; the silt acts as a filler to prevent the granular particles from rocking; while the clay serves not only as a binder but also provides pores minute enough to cause connecting moisture films which are supposed to have high cohesive properties¹.

It will be recognized that the clay is an important ingredient both when wet and dry, and should not show excessive expansion when wet, nor correspondingly high shrinkage in drying, as otherwise there is danger of unseating the nonplastic particles.

Such a stabilized road, although an improvement on one with a floating surface, will in course of time be subject to wear, and become dusty when the clay dries out and breaks up. It has been found that if some salts are added to the road mixture they tend to maintain the soil moisture and prevent dustiness of the road or raveling of the road surface.

In addition to this, it is claimed that these dissolved salts cause a thinning of the moisture films on the particles, thus permitting greater compaction of the road mixture, and thereby increasing its density. This fact can be proved by noting the increased compaction which a road mixture shows when certain salts are added to the soil water.

Since certain chemical substances such as sodium chloride are electrolytes, the question arises whether they cannot have some effect on the qualities of the clay in the soil mortar, and perhaps improve them.

In view of the fact that certain salts seem to exert a favorable effect on the road mixture it is desirable that they be held in the road soil as long a time as possible.

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¹ *Public Roads* (Feb. 1935) **15**, 275.

Whatever role the added compound plays in road stabilization must be largely because of its effect on the behavior of the clay, although it should be understood that much of the beneficial effect may be lost if the road ingredients are not properly proportioned.

One of the chemicals recently widely used to increase the permanence of a stabilized road is sodium chloride (common salt), a number of highways having been treated with this material. Comparatively little has been published regarding the use of this chemical in road mixtures, but its use as a dust layer has been referred to from time to time, and the hard surfaces of some dry lakes in the arid region of the west afford a good example of the binding power of this material. Instances are also known where clay-gravel roads have been sprinkled with sea water to give them a hard surface.

Within the past year the writer has had occasion to carry on a series of tests to determine the behavior of salt on stabilized road mixtures, the results of which seem to be worth recording. Since clay is the main bonding constituent in such a road, the experiments were directed chiefly towards determining the effect of the salt on this material.

The clays selected represented several different types of varying plasticity, one of them being highly calcareous. They may therefore be said to represent the range of material used in road construction. Each clay was mixed with 1, 3 and 6 per cent salt, and tested for plasticity index², shrinkage limit, field moisture content, volume shrinkage, water retention, and permeability to water.

The average figures for a "statistical soil" as published by the Bureau of Public Roads³ show that the plasticity index and field moisture equivalent increase with the clay content, while the shrinkage limit decreases. Thus, for example, a statistical soil with a clay content of 50 would have a liquid limit of 69, and a plasticity index of 34, a field moisture content of 37 and a shrinkage limit of 12. If a soil has a higher field moisture equivalent than a statistical soil it is an indication of higher expansive qualities. If field moisture equivalent is equal to or lower than shrinkage limit it indicates inappreciable shrinkage qualities.

Tests show that if sodium chloride is added to a clay its general effect seems to be to decrease the volume shrinkage, increase the shrinkage limit slightly, but cause little change in the percentage of field moisture equivalent. This, then, has the effect of bringing these two values closer together, which is an advantage, because it means less expansion of the clay when it is wet and less shrinkage on drying. The plastic index of salt-treated clay is lowered somewhat, but not to a detrimental degree.

A slight amount of moisture in the form of thin films surrounding the soil fines is desirable, as in this condition the cohesiveness between the clay

² For explanation of these terms, see *Public Roads* (Oct., 1931) 12.

³ *Public Roads* (March, 1933) 14.

and sand is increased. The addition of certain soluble salts, therefore, in addition to encouraging greater compaction of the road, assists in retaining the moisture whether it is concentrated on the surface or in the underlying road soil. The retentive power of salt for moisture when mixed with clay can be seen from the figures in Table 1.

TABLE 1.—*Behavior of Salt Mixed with a Calcareous Glacial Clay*

Salt, per cent.....	0	3	6
Water lost in 24 hr. air-drying, per cent.....	14.7	9.96	3.85
Water lost after additional 6 hr. drying at 105° C., per cent....	0.25	2.83	9.78

Table 2 represents the results obtained with a sandy clay containing 6 per cent sodium chloride. The samples were then allowed to dry for 120 hr. in air, and finally in an oven to constant weight. The results given in the table show the percentage of water retained, expressed in

TABLE 2.—*Behavior of Salt with Sandy Clay*

	Water Retained, Per Cent		
	Freshly Mixed	48 Hr. in Air	120 Hr. in Air
Clay alone.....	26.9	2.19	1.5
Clay + 6 per cent salt.....	24.1	7.20	2.2

terms of oven-dried clay. It will be observed that less water was required to mix the clay to the same consistency when salt was present, and that the salt-treated clay retained a larger amount of water.

In another series of tests, Table 3, three clays were mixed with water to a plastic mass and then allowed to dry to a constant temperature in air, and in atmospheres of different degrees of humidity. Where the percentage of moisture exceeds 100 per cent it indicates that water was absorbed from the atmosphere. Of course, if the clay were mixed with sand and gravel the percentage of moisture retained would not be as large.

CAPILLARY MOVEMENT OF SALT SOLUTION IN CLAY

Water flowing on a road surface will tend to penetrate it, unless the material is already saturated. Conversely, if the soil rests on a moist subsoil, it will tend to draw the water upward by capillarity, to replace moisture that may be evaporated from the surface. Soluble salts present in the soil may migrate with the moisture, but some may move more rapidly than others.

It seems obvious that if a stabilized road soil contains a soluble salt, the latter might eventually be leached out, unless the pores of the soil

close up and retard the penetration of water, or unless the salt is held there by adsorption or base exchange.

To test the upward movement of salt through a clay-sand mixture, some 3-in. cylinders containing mixtures consisting of equal weights of clay and sand were set up. The lower inch contained 6 and 12 per cent respectively of salt. The lower end of the column was in contact with water, and evaporation was free to take place from the upper end. In each case the salt had reached the top of the column in 24 hr. If in the same experiment the upper end of the cylinder is closed to prevent evaporation, the salt will disperse upward through the moisture films coating the grains. Somewhat different but more interesting results are obtained if the salt solution is allowed to percolate downward through a column of soil mixture.

TABLE 3.—*Percentage of Original Water Left in Clay at End of Drying at Room Temperature*

CLAY ALONE			
	1	2	3
In air.....	3.4	2.8	18.7
55 per cent humidity.....	67.6	57.2	72.3
76 per cent humidity.....	70.8	66.8	76.3
87 per cent humidity.....	83.0	74.4	87.9
CLAY AND 1 PER CENT SODIUM CHLORIDE			
In air.....	7.0	6.6	20.8
55 per cent humidity.....	64.5	75.4	76.5
76 per cent humidity.....	81.7	66.3	81.4
87 per cent humidity.....	81.2	91.4	92.0
CLAY AND 6 PER CENT SODIUM CHLORIDE			
In air.....	25.2	52.0	41.7
55 per cent humidity.....	104.0	110.0	104.0
76 per cent humidity.....	108.0	112.0	109.0
87 per cent humidity.....	112.0	114.0	108.0
Clay under 0.005 mm.....	44.65	44.5	58.10
Silt.....	47.35	43.0	31.43

If we make up columns of a stabilized road mixture, with 3 and 6 per cent of salt respectively, and allow water to percolate down through them, the percolation is rapid at first, and then slows up. As an indication of the slowness with which the sodium chloride percolates, reference

may be made to a series of percolation experiments made with four different clay mixtures consisting of 75 per cent of standard Ottawa sand and 25 per cent clay. Sodium chloride amounting to 6 per cent of the weight of the column was placed on top of each and distilled water fed in from the top. The filtrate was tested at intervals with silver nitrate for the presence of chloride, and the results are interesting (Table 4). These tests

TABLE 4.—*Percolation Tests on Mixtures of Sand and Clay*
75 Per Cent Sand, 25 Per Cent Clay, Salt 6 Per Cent Weight of Column

Clay	Gave Chlorine Test at End of	Water Percolated, C.c.
A	71 hr.	70
B	22 hr.	115
F	41½ hr.	50
E	39 hr.	15

indicate how slowly the clay is leached out, even when there is free drainage at the bottom of the column.

RELATION BETWEEN SODIUM CHLORIDE AND ROAD SOIL

It is of interest to speculate what happens to the sodium chloride when mixed with wet clay. Some of it may be adsorbed, but other portions of the sodium cations may replace cations of other bases in the clay particles.

Sodium chloride even in small amounts causes coagulation of a clay, but if the sodium chloride is nearly all removed, the clay becomes dispersed. This action can be demonstrated by treating a dispersed clay suspension with sodium chloride. As a result of coagulation it settles. If the clay is washed repeatedly to remove the sodium chloride, the clay once more becomes dispersed.

When water percolates through a salted road soil the salt is leached out of the upper part, and the clay grains that were coagulated become dispersed, closing the pores and retarding further downward movement of the water. If the water evaporates from the surface, the brine creeps up again, reoagulating the clay and crystallizing at the surface.

The retardation of percolation caused by the salt also tends to preserve it in the road, obviating the necessity of replenishing the supply repeatedly during the season.

The mixture used for the stabilized road may consist of: (1) bank-run gravel mixed with clay overburden; (2) coarse gravel from one pit and clay as well as sand from another; or (3) crushed stone or slag, sand and clay. With all mixtures it is necessary to see that the fines show the necessary physical properties, such as plasticity index, etc. The coarse

aggregate should also show the proper gradation to satisfy the requirements of the Bureau of Public Roads⁴.

About 2 lb. of salt per square yard is usually added to a 3-in. course. This road mixture with salt added is then harrowed, bladed wet, and rolled.

It is not usually necessary to make more than one application of salt a season.

DISCUSSION

(*G. R. Mansfield presiding*)

[This discussion refers also to the paper by C. D. Looker, beginning on page 423.]

E. F. BURCHARD,* Washington, D. C.—This is not a discussion; it is merely to mention an interesting fact concerning the use of salt on the roads, probably not intentionally, either. In Manila, in 1920, they were spraying the streets of the city with sea water in order to lay the dust. Is there sufficient salt in sea water to really benefit the road?

H. RIES.—Nothing has been said in either paper about the dust-laying qualities but it might have been added that salt does help to make the road less dusty, and even when it is dry a vehicle going over it at considerable speed causes little dust. Sea water was found to be very effective in Virginia for laying dust, at least so I have been told by a person who had personal contact with it.

A. R. CHAMBERS,† New Glasgow, N. S.—In the town of Pictou, Nova Scotia, sea water has been used on the streets for several years, and no attempt has been made to stabilize the road or to add sufficient clay to permit of proper stabilization. At the same time, a surface has been gradually built up, and it has been clean and has withstood traffic rather well. Of course, it is not properly mixed. A manager of a hardware store in the town drew my attention to the possible corrosive effect of the magnesium content of the sea water. He said that silver-plated or nickel-plated goods in his store showed little spots.

MEMBER.—Has there been any experience in treating straight clay roads of the gumbo prairie type with salt?

C. D. LOOKER.—We do not know of any straight clay roads that have been treated with salt. We recommend that the road be stabilized according to specifications before the salt is used.

J. R. THOENEN,‡ College Park, Md.—What would be the effect on a salt-stabilized road in a heavy rain? Would the salt not dissolve and wash away?

What would be the effect on automobile traffic? Would it rust the steel driving over salt roads?

C. D. LOOKER.—The salt does finally get leached out of the surface. I tested the road at Ithaca while it was raining hard. There was no salt on the surface after it had rained for six or eight hours. Some was washed to the curb, and some of it had

⁴ *Public Roads* (Feb., 1935) **15**, 276.

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‡ U. S. Bureau of Mines.

gone down beneath the surface, but after about $\frac{1}{8}$ in. of the road had become moist, we got the effect that Professor Ries spoke of in his paper—the clay swelled slightly and the clay in the surface dispersed, closed the pores and stopped further percolation of water downward. During that time there was salt present $\frac{1}{2}$ in. below the surface, and the material $\frac{1}{2}$ in. below the surface was comparatively dry. The fine material was not moist enough to permit water to be squeezed out of it between the fingers so the road had not absorbed an appreciable amount of moisture.

These roads dried off quickly after rain. They are closed surfaces that shed water remarkably well, but because of absorption and sinking of the water into the surface, and various other factors, the salt finally has to be replenished. We have not had to replenish salt more frequently than once a year on roads carrying lighter traffic. The proper time to replenish salt is when the road surface is moist in the spring, or during rainy weather.

We do not have any difficulty to speak of with corrosion of cars. Modern paints are not affected by salts, so that we need not be alarmed about the dulling of the paint. If the paint gets knocked off a car, corrosion will take place in the absence of salt. We have never noticed that the salt does permanent injury to the chassis or the fenders. Fenders will rust driving over ordinary wet gravel roads.

J. R. THOENEN.—I have read reports of a company in Canada regarding a piece of ordinary clay or dirt road in part of which was a section covered with a stabilized road mixture. That road went through the winter. The stabilized road was smooth and the original road was just what one would expect a clay road in winter to be.

A. R. CHAMBERS.—I presume Mr. Thoenen refers to the section at Elmsdale, N. S. It was laid in the fall with what would now be considered not very good clay, but it did resist traffic very well through a winter that made maintenance of a road very difficult. There were thaws followed by freezing periods and the truck traffic did break through the road on each end of the test section. The test section was not very long; it was only several hundred feet.

O. BOWLES,* Washington, D. C.—Mr. Looker referred to an artificial rubber made of salt, coal and limestone. I wonder if we can get some more information about that.

C. D. LOOKER.—That has been published; the work of Father Nieuwald and co-workers,⁶ of Notre Dame University, will bring you up to date on that. Acetylene is made from coal, limestone and water. Vinylacetylene is then made from acetylene using chlorine compounds as catalysts. Hydrogen chloride is added to vinylacetylene to make chloroprene, which is then permitted to polymerize to chloroprene rubber or "Duprene." The chlorine for making hydrogen chloride and chlorine compounds used as catalysts comes from salt. I have at home an eraser made of Duprene, made by the duPont Company. Recent developments in Germany are along similar lines; synthetic rubbers have been made which give great promise of being applicable for everyday automobile use. It is said that if we had to we could make all the rubber we need synthetically.

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⁶ *Jnl. Amer. Chem. Soc.* (1931) **53**, 4197, 4203.

Carbon Dioxide Accumulations in Geologic Structures

BY J. CHARLES MILLER,* MEMBER A.I.M.E.

(New York Meeting, February, 1937)

NATURAL carbon dioxide has recently been exploited in the United States in consequence of oil and gas developments in the Western States and the growing demand by transcontinental and transoceanic shippers for solid carbon dioxide. The discovery by drilling of commercial deposits resulted in consideration of its geographic and geologic distribution and occurrence. Its wide distribution in nature is evidenced by its occurrence in the atmosphere, hydrosphere and lithosphere. The question of origin of carbon dioxide involves consideration of theories similar to those concerning natural hydrocarbons, although available evidence favors an inorganic rather than an organic source. Areas favorable to development of commercial production appear to be those in proximity to igneous rocks or in the vicinity of decadent volcanic activity, where, it has long been known, the gas is evolved either free or dissolved in the waters of fumaroles, geysers or mud pots. Structure contour maps are presented showing the Wagon Mound anticline in New Mexico and the Farnham anticline, Utah. A well drilled on the latter supplies gas to a 20-ton plant manufacturing solid carbon dioxide at Wellington, Utah, on the Denver and Rio Grande R.R., whence it is shipped to surrounding territory and to the West Coast. Before funds are invested in a plant for the manufacture of liquid or solid carbon dioxide, whether supplied by wells or springs, an effort to ascertain the volume of gas available is advisable. The use of present methods of estimation of gas reserves with modifications to account for the quantity absorbed by any water present in the producing formation is recommended. The Farnham anticline is used as an example and computations are made of the probable volume of gas originally present in the structure. No attempt was made to compute reserves in the Niland area, California, for the reason that in such areas it is believed the volume is indeterminate.

INTRODUCTION

Carbon dioxide, in liquid and solid form, has found a wide variety of uses other than its primary application in carbonating beverages. It is now used in more than 40 different ways, ranging from cooling or freezing to use in gas or liquid form for mechanical purposes, chemical industry, explosive, fire extinction, and tire manufacturing. Undoubtedly the chief supply of carbon dioxide is obtained from industrial plants or artificial sources and until recently natural deposits of the gas have received little consideration. Few will question the utility and perhaps dependability of artificial sources, which possess the advantage over

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many natural deposits of being situated at or near the point of consumption. This advantage of location pertains chiefly to the solid, which evaporates continuously, consequently long storage and shipment are not conducive to economic marketing. In some of the Western States, and perhaps in some foreign countries where natural sources or accumulations may be present, and where coal or by-product gas is not available, consideration of geological conditions requisite to commercial deposits provides a means of determining the feasibility of exploiting such occurrences.

HISTORICAL REVIEW

The presence of carbon dioxide in the free or combined state in springs has long been recognized, and perhaps the first mineral springs to attract the American colonists were those at Saratoga, N. Y., which according to Kemp²⁰ were known as early as 1767. These waters usually issue from the wells or springs at temperatures between 50° and 54° F., slightly above the annual mean temperature. The natural flow of the springs was utilized until about 1870, when the first well was drilled. During the period between 1892 and 1905, when it was realized that the carbon dioxide in the mineral waters could be extracted profitably, more wells were drilled. Competition between the producers of carbon dioxide resulted in excessive drilling and pumping of wells, so that between 1910 and 1912, before pumping was stopped, approximately 400,000 gal. of mineral water⁴ was pumped daily. The water level is reported to have been lowered 100 to 150 ft. by this overdraft. The gas is found at depths of 150 to 600 ft. and at the flush period of production 30 or more wells contributed to the supply. During the most active period, between 1904 and 1908, the annual production of carbon dioxide ranged between 4,000,000 and 5,000,000 lb. per annum. In 1927 the State received royalties totaling about \$15,000 from gas produced on the State Reservation.

Until the creation of the State Reservation at Saratoga and legal restraint from pumping in 1908, the compression of carbonic acid gas was an important industry at Saratoga Springs and in the vicinity. This was one of the first important natural occurrences to be used exclusively for carbonic acid production, and undoubtedly it demonstrated the feasibility of utilizing similar supplies. Nevertheless, a number of years elapsed before the natural gas from wells again attracted attention.

In 1926 the Continental Oil Co. completed a well on the McCallum anticline, near Walden, Colo., in the NW. $\frac{1}{4}$ NW. $\frac{1}{4}$ of sec. 12, T.9N., R.79W., 6th P.M., which produced, according to reliable estimates, 50,000,000 cu. ft. of carbon dioxide gas with a few barrels of oil of 45.8°

²⁰ References are at the end of the paper.

A.P.I. gravity. An experimental plant for the manufacture of solid carbon dioxide was constructed in the field in 1930 but this was dismantled in the summer of 1932. This discovery appeared to stimulate interest in natural carbon dioxide from wells, and in June 1930 the Carbon Dioxide and Chemical Co. completed a carbon dioxide well on the Farnham anticline in the SW. $\frac{1}{4}$ of sec. 12, T.15S., R.11E., S.L.M., Utah. In 1931 the Estancia Valley Carbon Dioxide Development Co. commenced drilling a well for carbon dioxide in the SE. $\frac{1}{4}$ SE. $\frac{1}{4}$ of sec. 12, T.7N., R.7E., N.M.M., Wilcox anticline near Estancia, N. M. Late in 1932 the Salton Sea Chemical Products Corporation commenced drilling a well for carbon dioxide near Niland, in Imperial Valley, Calif., in the SW. $\frac{1}{4}$ of sec. 28, T.11S., R.13E., S.B.M. Plants for the manufacture of solid carbon dioxide were subsequently constructed in each of the last three localities named.

DISTRIBUTION IN THE UNITED STATES

Geographic.—An interesting table by Dr. Peale²⁹ illustrates the frequency of mineral springs in the United States and discloses a preponderance of such springs in the western United States. Peale states that there is a far greater prevalence of thermal springs in the West than in other sections, and that many of them are carbonated. Analyses of their waters often reveals an abundance of carbon dioxide either in the free or the combined state. The geysers and springs of Yellowstone Park are known to contain large volumes of the gas. It is worthy of note that Rogers (ref. 32, p. 119) reported analyses of gas from springs in France ranging from as much as 41.5 to 99.8 per cent carbon dioxide. An old fumarole at Agnano, near Naples, Italy, representing Vesuvius gas, and that emitted by a hot spring in Iceland he reports contain 96.52 per cent and 79.5 per cent carbon dioxide, respectively. It is generally recognized that natural gases from wells in this country containing noteworthy amounts of carbon dioxide are confined chiefly to the western United States. Lang²¹ discussed two of the early discoveries of carbon dioxide gas in wells drilled in New Mexico and probably was the first to suggest a causal relationship between these large quantities of the gas and the results of igneous activity in northeastern New Mexico. Dobbin¹³ discussed the occurrence and origin of carbon dioxide gas in the United States and other countries, citing wells in fields or areas in Montana, Colorado, New Mexico, Utah and California. The wells near Niland, Calif., at the southern end of the Salton Sea, which supply gas to two plants in that vicinity, were discussed in a previous paper.²⁷ Rogers states (ref. 32, p. 37) that some natural gas produced in California contains unusually high percentages of carbon dioxide, carrying "as much as 49 per cent of carbon dioxide, and a great many contain 10 or 15 per

cent . . . but true natural gas containing noteworthy proportions of this constituent occurs almost wholly in California."

Geologic.—Carbon dioxide has been discovered in the United States in wells drilled into rocks ranging in age from Cambrian to Tertiary. Its source in springs is not easily determined but an exhaustive investigation of the geysers and springs and of boreholes in Yellowstone Park by Fenner,¹⁴ Allen, Day and Merwin² has thrown much light on the possible age and character of its source rocks. Its wide distribution or occurrence is related to its function in natural processes.

OCCURRENCE

Carbon dioxide is present in the free or combined state in the atmosphere, hydrosphere and lithosphere. Commercial accumulations are confined largely to the lithosphere, and the waters and gases issuing from springs. Some, however, becomes trapped in geologic structures and may be obtained only by drilling. These are at present the chief natural sources of commercial interest. Clarke^{9,10} has considered carbon dioxide in his *Data of Geochemistry* and in the present paper frequent reference has been made to that publication and to the literature that he reviewed.

Atmosphere.—The present atmosphere has little bearing on commercial accumulation of the gas but its origin is intimately connected with the origin of the globe. For this reason the discussion of the atmosphere has been included.

Air consists roughly of four-fifths nitrogen and one-fifth oxygen, but carbon dioxide content is very nearly uniform and normally present in the atmosphere to the extent of three volumes in 10,000. If the rate of generation of carbon dioxide by combustion of coal and other substances were constant and no disturbing factors interfered, the amount of it in the atmosphere would be vastly increased over a period of time. The large factors tending to maintain a balance are: (1) the decomposition of carbon dioxide by plants, with liberation of oxygen, (2) the consumption of carbon dioxide in the weathering of rocks, and (3) absorption by surface waters. The liberation of oxygen began with the appearance of low forms of plant life, possibly reached a maximum during the Carboniferous era, and since has diminished. Clarke points out, however, that two opposite views of the composition of the primitive atmosphere have been taken: (1) that it was rich in oxygen, and (2) that it was oxygen free. He further states (ref. 9, p. 57) that:

The balance or lack of balance between carbon and oxygen is, after all, only one factor in the problem. The origin of the atmosphere as a whole is a much larger question and our answers to it must depend upon our views as to the genesis of the solar system. If we accept the nebular hypothesis, we are likely to conclude that the atmosphere is merely a residuum of uncombined gases which were left behind when the globe assumed its solid form. That seems to be the prevalent opinion, although it must be modified by the observed facts of volcanism.

In regard to the primitive atmosphere Clarke says (ref. 9, p. 55):

The accumulations of carbon in the lithosphere such as the coal measures, the limestones, and the like, have led some geologists to assume that the atmosphere at some former time was vastly richer in carbonic acid than it is now; but the fossil records of life suggest that the differences could not have been extreme. With a large excess of carbon dioxide, the existence of air-breathing animals would be impossible. Only anaerobic organisms could live. It is clear that the stored carbon of the sedimentary rocks was once largely in the atmosphere but was it ever all present there at any one time?

Such a supposition is improbable. The known carbon of this lithosphere, if converted into carbon dioxide, would yield nearly 25 times the present mass of the entire atmosphere and the atmospheric pressure at the surface of the earth would be enormously increased.

Several theories have been advanced to account for the source of carbon dioxide in the atmosphere by Hunt,¹⁷ Meunier,²⁶ and others who postulated either a cosmic or a volcanic source, but Clarke holds that "A cosmical atmosphere has no assignable limit; an atmosphere of volcanic origin must sooner or later be exhausted."

Another theory relative to the atmosphere, favored in varying degrees by Koene, Phipson, Lemberg, Stevenson and Kelvin, is based upon the belief that the unoxidized but oxidizable substances in primitive rocks are sufficient in quantity to absorb all oxygen of the air.

The effect of atmospheric carbon dioxide and that in the free or combined state in the ocean, therefore, has an important bearing on discussions of theories of origin of carbon dioxide in geologic structures. In certain theories of climatic changes extraordinary importance is attached to variations in the amount of carbon dioxide in the atmosphere. Periods during which the carbon dioxide content of the atmosphere was high are considered to have been uniformly warm, those in which it was low are considered to have been times of cold weather or glacial periods. Brooks⁵ concludes that carbon dioxide can never have been an important factor in climatic variations. He admits, however, that there appears little doubt that variations in the quantity of carbon dioxide in the atmosphere during different geological periods were considerable. Bunsen's examination of air from rain water at different temperatures showed that it contains as high as 2.92 per cent CO₂ at 0° C.

Hydrosphere.—A river or fresh-water lake owes its composition to the streams or ground waters contributing to it, and, a river particularly, to the conditions existing in the terrane along its course. Carbonic acid of either atmospheric or organic origin is the most abundant of chemical agents in the lake or river waters. Hence carbonate waters are the most common and as the streams form the great continental rivers the carbonate type becomes more pronounced. When these river waters reach the sea the calcium carbonate may be deposited as limestone or calcareous shale, or it may be withdrawn as such by living animals to form shells.

In addition to carbonic acid, other dissolved gases in ocean water are nitrogen and oxygen from the air. The oxygen maintains life of the marine organisms and then oxidizes the dead matter of organic origin. Free or half combined carbonic acid may be absorbed directly from the atmosphere, or brought down in rain; or from submarine volcanic springs. The carbonic acid content of vadose water or water falling on the surface as rain is too small to have any effect on accumulation at depth. The acid, however, may be liberated by chemical or thermal action of waters of deep-seated origin on limestone, dolomite or other rocks containing lime.

It has been estimated that the ocean contains from 18 to 27 times as much carbon dioxide as the atmosphere and it is considered the great regulative reservoir of the gas. This fact has given credence to the assumption that the gas may have been present in connate waters.

The waters from mineral wells or springs may be classed according to their negative radicles as chloride, sulphate, carbonate or acid waters. The carbonates of calcium, magnesium, and iron are easily dissociated, giving carbon dioxide. Carbonate waters or waters of volcanic origin are of chief concern. The most notable springs or geysers of volcanic origin in the United States are those of Yellowstone Park. Allen, Day and Merwin² found that 7 out of the 40 Yellowstone gases analyzed contained less than 80 per cent carbon dioxide. Next to steam and carbon dioxide, hydrogen sulphide is the most important of the hot springs gases in Yellowstone. In addition to these, spring gases of volcanic origin may contain hydrogen, marsh gas, nitrogen and argon. Allen and Day calculated the composition of the uncontaminated permanent gases or original magmatic gas in the Yellowstone Park area as follows: CO_2 , 98.26; H_2O , 0.11; CH_4 , 0.11; $\text{N}_2 + \text{A}$, 0.86 and H_2S , 0.66. These investigators found that the alkaline springs gave a very small amount of gas, compared with other springs. Such springs are conspicuously high in nitrogen and methane. The bicarbonates of the alkaline springs are formed by the "absorption of magmatic carbon dioxide in chemical reactions below ground" (ref. 2, p. 510) thus reducing the volume of gas that escapes at the surface. In the acid or sulphate areas, on the other hand, abundant gases escape from the springs which contain little or no bicarbonate. Another difference between gases of sulphate and alkaline areas is the almost complete absence of hydrogen sulphide in the latter. Mud pots or mud volcanoes are simply extreme cases of the sulphate type of hot springs, the mud of which is composed of insoluble products of rock leached by acid water.

Lithosphere.—The effects of the dissolved volcanic gases on the rocks with which they come in contact are the formation of clays, replacement of soda and lime by potash and an increase in silica. If we assume calcium

bicarbonate in solution, the changes due to escape of gas may be described according to the following formula:



The result is the deposition of calcareous sinter. Clarke (ref. 9, p. 34) gives the weighted average composition of the lithosphere and calculates that CO_2 is present in combination with igneous rocks to the extent of 0.101 per cent, with shales 2.63 per cent, sandstone 5.03 per cent, limestone 41.54 per cent, or for a weighted average of igneous and sedimentary rocks to the extent of 0.35 per cent.

Bituminous shales and similar materials when heated give the products of dry distillation, such as hydrocarbons, especially marsh gas, carbon dioxide, ammonia and tarry matter. Limestone under similar conditions, that is, in the presence of intrusives, would yield carbon dioxide. However, these gases are common volcanic emanations occurring in many places where there is no positive evidence of the presence of limestone, sulphide deposits or organic materials.

Chamberlin⁷ classified analyses of gases in igneous rocks according to age and found in his examination of rock samples heated in vacuo that there was a greater volume of gases in Archean rocks. He also noted "a rapid and steady decline in the quantity of every gas, in passing down the columns from the Archean through the Proterozoic and Tertiary to recent lavas." He assigns these differences to a combination of causes—older rocks may yield more gas than the recent, owing to metasomatic changes that have been slowly taking place within the rocks, or the early magmas may have been more highly charged with gas.

Analyses collected by Allen (ref. 1, pp. 70–80) show the percentages of CO_2 and other gases taken from different volcanic sources throughout the world. Although these analyses have little or no bearing on the quantity of the gas contained in igneous rock, they show the importance of carbon dioxide as a constituent of volcanic gases. It is interesting to note the percentages of carbon dioxide present in the volcanic gases selected from widely separated areas as given by Allen (Table 1).

TABLE 1.—*Carbon Dioxide in Volcanic Gases*

Location	Date of Analysis	CO_2 , Per Cent
Hawaii: The crater.....	1912	73.9
Sicily: Paterno spring of Vallancella.....	1865	99.78
Colombia: Purace solfatara.....	1868	98.2
Alaska: Katmai Nova Rupta Basin.....	1919	72.0
Canary Islands: Pico de Teyde north fumarole.....	1907	70.8
Java: Papandajan fumarole.....	1908	16.8 to 24.00

In sedimentary and metasedimentary rocks, Chamberlin noted that age appears to make less difference in the gas evolved, although in these rocks, as in igneous rocks, carbon dioxide was present in large percentages.

He further noted from the analyses of gases from igneous rocks classified according to granularity that the fine-grained rocks yielded the largest quantity of gas. Gases in meteorites analyzed showed that carbon dioxide was more abundant in stony specimens than in iron meteorites. Besides carbon dioxide, occluded gases may occur in various rocks or minerals, and such gaseous inclusions contain quantities ranging from a trace to 98 per cent of hydrogen sulphide, sulphur dioxide, ammonia, nitrogen, marsh gas, methane, oxygen, argon and helium. Chamberlin⁷ observes that "there is reason to suppose that at the depths where lavas originate hydrogen and oxygen exist combined as water, since up to temperatures of 2000° C. the dissociation of water takes place only to a limited extent."

An examination of available data on volcanic eruptions led Clarke (ref. 10, p. 292) to conclude that the certain regular order of appearance of volcanic gases is due to succeeding chemical changes and is essentially that laid down by Deville and Leblanc, except that the early evolution of hydrogen and carbonic oxide is taken into account as follows:

First. The gases issue from an active crater at so high a temperature that they are practically dry. They contain superheated steam, hydrogen, carbon monoxide, methane, the vapor of metallic chlorides and other substances of minor importance.

Second. The hydrogen burns to form more water vapor, and the carbon gases oxidize to carbon dioxide. From the sulphur, sulphur dioxide is produced. The steam reacts upon a part of the metallic chlorides, generates hydrochloric acid, and so acid fumaroles make their appearance.

Third. The acid gases of the second phase force their way through the crevices in the lava and the adjacent rocks; their acid contents are consumed in effecting various pneumatolytic reactions. The rocks are corroded, and where sulphides occur hydrogen sulphide is set free. If carbonate rocks are encountered carbon dioxide is also liberated.

Fourth. Only steam with some carbon dioxide remains and even the latter compounds soon disappear.

These phases have been similarly described by others, including Delkeskamp,¹² who assigns a temperature of 1000° C. to the first phase, 600° to 100° C. to the second, and 100° to 50° C. to the third phase.

The statement is generally accepted that water vapor or steam forms 99 per cent of the entire gaseous output of volcanoes, particularly during the spectacular phase, but emanations of carbon dioxide with nitrogen and oxygen mark the last phase of volcanic activity.

ORIGIN

The occurrence of carbon dioxide is intimately connected with questions of its origin and accumulation in geologic structures; therefore any attempt to determine the origin of carbon dioxide in a particular locality leads to consideration of many possible sources of the gas and to realization of the complex factors that may contribute to its accumulation in commercial amounts. Two hypotheses of origin may be advanced to explain its source: (1) organic or sedimentary, and (2) inorganic or volcanic. To this some would add a third source, which may be related directly to the primitive atmosphere, but this is not considered an adequate source for commercial deposits.

Organic.—Carbon dioxide is derived largely from organic material consisting of the remains of planktonic organisms present in the water above the sediment and from other vegetable or animal matter contributed to the sea, the decay of which may be assisted by bacteria. During the decay of seaweeds, according to F. C. Phillips,³⁰ a little methane is at first evolved, but the generated gases consist largely of carbon dioxide, hydrogen and nitrogen. Organic matter may be deposited with the producing sand or in the adjacent beds. Retardation of organic decay by diminution of the supply of oxygen results in the formation of carbon dioxide from carbonaceous matter and the suppression of biochemical decomposition by diminution of the supply of oxygen may be caused by rapid deposition of sediments or by deposition beneath stagnant water. However, Trask³⁶ points out that the oxygen content decreases progressively with depth in buried sediments and the original organic source material decreases very greatly in quantity before it reaches the sea floor and is deposited in sediments. If the sea in which the sediments were deposited contained a high concentration of carbon dioxide and the atmospheric content at the time of deposition was relatively high, connate waters resulting therefrom may contain a relatively high amount of carbon dioxide. Under ordinary conditions the amount in sea water is slightly below that required for equilibrium with atmospheric carbon dioxide, and this is probably less than 0.1 gram per liter at ordinary temperature, according to Wells.³⁸ Other investigators have reported 0.023 per cent at 20° C. to 0.044 per cent at 0° C. Beyond a depth of 2500 fathoms, little calcium carbonate is found in sea water. The amount of free carbonic acid, however, increases slightly with depth. The origin of organic carbon dioxide is, therefore, closely allied to that of petroleum. Rogers³² believes that much of the carbon dioxide in the California gas, for example, has probably been formed by oxidation of the hydrocarbons through contact with mineralized waters. The existence of large volumes of carbon dioxide in association with commercial accumulations of petroleum is further exemplified by the developments.

near Walden, Jackson County, Colorado, which were discussed in a previous paper.²⁸

Inorganic.—The presence of carbon dioxide and ammonia on the planets has been determined by astrophysicists, and consideration of the possible sources of these gases in the earth, whether they be of cosmic, volcanic or organic origin is an involved subject. Moreover, carbon dioxide, ammonia and the sulphur gases are of too general occurrence to be connected invariably with the presence of bituminous shale, limestones, and comparatively superficial formations, for the local existence of which there is often little or no evidence. It is admitted, however, that these gases may find occasional access to the conduits of volcanic gases from outside sources. Allen (ref. 1, pp. 29–80) calculated the average composition of gases in igneous rocks based on Chamberlin's table (ref. 7, p. 27) of average volumes of various gases obtained from the igneous rocks per volume of rock, reduced to 0° and 1 atmosphere. Allen's Table 4 is given here as Table 2.

TABLE 2.—*Average Composition of the Gases in Igneous Rocks*
Calculated in Volume Percentages

Order	Type of Rock	Num- ber of Analy- ses	Composition, Per Cent						
			H ₂ S	CO ₂	CO	CH ₄	H ₂	N ₂	Total
1	Basic schists.....	2	0.0	51.6	2.4	0.6	43.7	1.7	100
2	Diabases and basalts...	14	2.6	53.8	6.0	1.6	34.5	1.5	100
3	Gabbros and diorites...	11	0.4	48.8	2.7	1.5	44.2	2.4	100
4	Granites and gneisses...	19	0.0	46.0	6.9	1.6	42.6	2.9	100
5	Andesites.....	7	0.0	77.8	7.5	2.5	8.4	3.8	100
6	Syenites.....	4	0.0	14.4	5.6	4.0	72.8	3.2	100
7	Rhyolites.....	4	0.0	79.3	5.7	2.3	6.9	5.8	100
8	Miscellaneous porphyries	2	0.0	40.5	7.6	5.1	41.8	5.0	100

Where the depth of burial of granitic masses is sufficient to reach a temperature of 400° to 500° C., carbon dioxide could be expelled at such points when the pressure of the gas is great enough to escape to points of reduced pressure if open fractures, crevices, or other openings exist. Van Orstrand has estimated³⁷ that this temperature is generally attained at depths of less than 12.4 miles. If a temperature gradient is 1° C. per 100 ft., the depth necessary to attain this temperature would be 36,000 ft., at which approximate depth no crevices or open fractures are likely to exist at present. It is worthy of note, however, that igneous intrusives, such as dikes, necks and other feeders or roots, contract upon cooling, and unless movement of strata occurs, surface waters may gain entrance to great depths, possibly 10,000 ft. or more. Spicer's recent computations³⁵ show that the boiling temperature of water is reached in one-third of the localities represented at a computed depth of 7000 ft. or less, and

at two-thirds of the locations the depth to this temperature is 10,000 ft. or less. By extrapolation of the temperature curves for a few localities given in his table, he has computed that the critical temperature of water, 689° F., is reached at depths less than 30,000 ft. where temperatures of 212° F. are likely to occur at depths of 7000 ft. or less. Where a temperature of 212° F. is possible at 10,000 ft. or less, the critical temperature is with few exceptions reached at depths of less than 40,000 ft. In connection with an igneous mass at relatively shallow depths within the zone of circulating waters, evidence cited shows that volcanic carbon dioxide is added to the waters. Connate water or fossil sea water trapped with the sediments may contain carbon dioxide in free or combined state but in a relatively small quantity. Concentration of the gas in suitable

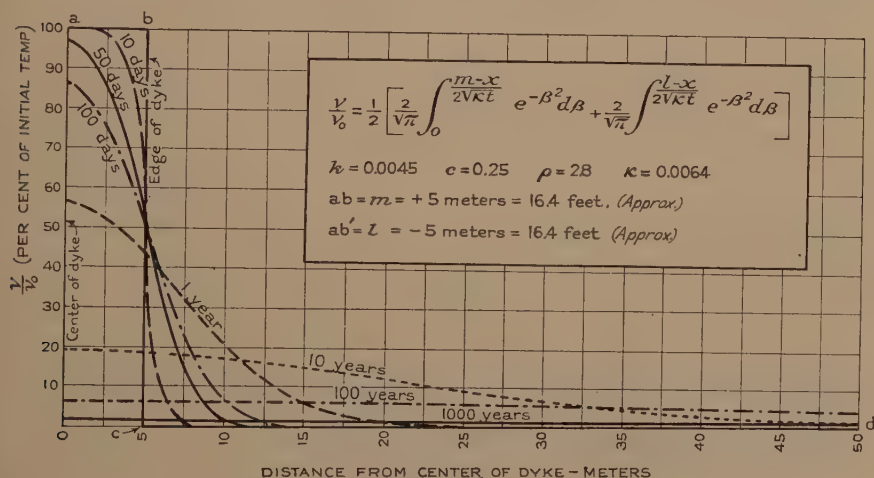


FIG. 1.—FLOW OF HEAT FROM DIKE, BY C. E. VAN ORSTRAND.

traps is a necessary condition to accumulation of commercial supplies of carbon dioxide, as well as its separation from water by specific gravity.

A curve (Fig. 1) showing the rate of flow of heat from a 32-ft. dike into adjacent rock of similar thermal conductivity at a temperature of 0° C. was calculated by C. E. Van Orstrand, of the Geological Survey. The mathematics of this curve is based on equation 25 by Ingersoll and Zobel.¹⁸ The figures on the left side of the graph represent the percentage of the initial temperature of the dike, those at the bottom show the distance (in meters) from the center of the dike to which the heat penetrates the wall rock in a given time. It is evident that if the temperature of the intruded dike material is 1300° C. about 50 per cent of this temperature, or 650° C., would result at the contact immediately after intrusion; further, that at the end of one year following intrusion its heat would affect the rock 65 ft. from the edge of the dike, where a temperature of about 26° C. might be recorded. At the end of one year the flow of heat would

not have affected strata 75 ft. or more distant from the edge of the dike. The curve is introduced to show the extreme slowness with which heat flows from a molten body and the improbability of heat from a dike causing any marked increase in temperature in the country rock sufficient to result in evolution of carbon dioxide contained therein, except in the immediate vicinity of the intruded mass. If the rocks adjacent to the dike are fused so as to close the voids, the emission of carbon dioxide and other gases from the magma would be retarded or prevented at least temporarily. The effect of such intrusions on a coal bed would result only in the evolution of volatile matter and the formation of coke. If the dike penetrates limestone or other limy beds the formation of carbon dioxide by reduction of carbonates might also be effected. The influence of the overburden on the formation of carbon dioxide is to be considered in this connection.

Commercial experience in the calcination of limestone, and the experimental data by Le Chatelier, reveal that the harder, denser and more crystalline the calcium carbonate is, the slower and more reluctantly will the carbon dioxide be evolved. The presence of silica or aluminum silicate accelerates the decomposition. Le Chatelier²² gives the dissociation temperature of calcium carbonate as 812° C. at atmospheric pressure. As the pressure is increased the temperature required also increases and vice versa. Lovering (ref. 23, pp. 87-100) points out that:

The rate at which heat will move from an arrested magma into the country rock adjoining it depends on (1) the initial temperature distribution; (2) the time elapsed after intrusion; (3) the specific heats, densities and thermal conductivities of the intrusive and of the country rock; (4) the presence of endothermic or exothermic reactions; (5) loss of heat through gaseous transfer; (6) the shape of the body.

From Lovering's curves (ref. 23, p. 99), his Plate 2, based on data from International Critical Tables, and from Ingersoll and Zobel, shown in

TABLE 3.—*Heat Imparted to Beds Immediately after Intrusion by Basalt Dike at 1300° C. (2372° F.)*

	K_1/K_2	θ_{ca}	Maximum Temperature Wall Rock	
			Centigrade	Fahrenheit
Coal.....	2.1229	0.68	884	1623
Shale.....	1.2666	0.55	715	1319
Limestone.....	0.7293	0.42	546	1015
Coral limestone.....	0.8755	0.47	611	1132
Gritty limestone.....	1.2500	0.57	741	1366
Oolitic limestone.....	0.8837	0.47	611	1132
Dolomite.....	0.7524	0.43	559	1038
Sandstone.....	0.7916	0.44	572	1062

his Table 1, the accompanying Table 3, giving the maximum temperature induced in beds of dissimilar composition, is computed for the special case of a dike at 1300°C . intruded in the strata listed. It is interesting to note the heat transfer that might result when a dike penetrates successive beds similar to those tabulated. The constant K is the ratio of the conductivity to the square root of the diffusivity. K_1 is the constant for the basalt and K_2 that for the rock penetrated. The ratio K_1/K_2 determines the temperature at the contact during the period immediately following intrusion. The symbol θ_{ct} represents the proportion or percentage of the initial difference in temperature between wall rock and intrusive, assuming the temperature of the wall rock to be zero.

If the temperature of dissociation of calcium carbonate is 812°C . or higher, depending on conditions, the table indicates that no carbon dioxide gas would be evolved from the lime rocks adjacent to the dike. With a sill, similar conditions may exist but the lava sheet probably has no bearing on commercial accumulations of carbon dioxide, as such gas formed by combustion of coal or organic matter would be lost to the atmosphere. Its effect, however, would be reflected by the temperatures of subsurface rocks within the limits indicated by the table. Experiments show that below 400°C . carbon dioxide is the principal gas evolved from rock material heated in vacuo. These temperatures, therefore, probably would be sufficient to expel occluded gases from the rocks affected. The rate of heat flow and distance from the intrusion are to be considered in this connection.

If this is a true interpretation of conditions in the geologic past, the indirect or thermal effect of volcanism on sedimentary deposits similar to those considered would result in little or no production of gas by chemical or physical change in the beds intruded. Nevertheless, the presence of steam in the magma might establish conditions favorable to generation of carbon dioxide from organic matter, or reduction of minerals.

Further evidence favoring an igneous source for carbon dioxide accumulations is furnished by the data computed by Allen (ref. 1, p. 29) for gases expelled from various types of igneous rocks measured at 100°C . and one atmosphere pressure. It was found that the gases from the average granite would occupy a volume 30 times as great as that of the rock from which they were expelled and the gases from the average basalt would amount to a volume 87 times that of the rock itself. If a corresponding emission of gas occurred in igneous intrusions generally, the resulting gas under pressure would find access to crevices and porous formations in proximity to such intrusions.

The critical temperature of carbon dioxide is about 88°F . and the critical pressure is 1070 lb. per sq. in. Unless a very low annual mean temperature and a pressure greater than 1070 lb. exist in a particular locality, it seems logical to conclude that carbon dioxide in commercial

quantities exists as a gas in the pores of the reservoir except that portion absorbed by any water that may be present.

FAVORABLE AREAS

Available information indicates, therefore, that carbon dioxide accumulations of either organic or inorganic origin are possible, and available evidence favors the inorganic. In areas where it is of organic origin, the distribution, quantity and quality of source rocks would govern the size and extent of commercial accumulations in reservoir rocks in folds or geologic structures of similar effect, assuming that the physical principles governing its accumulation are the same as those governing hydrocarbons. The location in the geologic past of buried shore lines, inland seas, or lakes, is then an important factor in the selection of structures worthy of prospecting.

Carbonated springs, mud pots and some thermal springs or geyser areas representing the remnants of volcanic or subvolcanic activity are likewise areas to be considered as favorable for well locations. While such springs may be used directly, the drilling of wells nearby affords a means of avoiding the escape of free or dissolved gases to the atmosphere. At Saratoga Springs, N. Y., the wells are in basins near springs and the waters ascend partly under hydrostatic head. In thermal spring or mud-pot areas the accumulation or the supply of magmatic gases or liquids is governed more or less by proximity to the buried igneous mass. While the dome-shaped surface of a laccolith or the effect of intrusion of a volcanic neck or a dike may be similar to the folds in other regions, it may provide favorable conditions for accumulation under hydrostatic, volumetric, or capillary control as defined by Herold.¹⁵ Sealed faults and lensing of the reservoir rocks may result in favorable conditions for accumulations also. Faulting, however, may produce suitable channels for migration of gases or liquids into porous formations.

The fact that carbon dioxide in commercial quantities occurs in anticlines or domes has been demonstrated by drilling. Whether these folds are the effect of deep-seated igneous intrusions, compressive forces, or differential settling of the sediments over an old land mass has not been determined. However, in one of the areas discussed, Wagon Mound, N. M., there is no doubt of igneous activity after the sediments were laid down, and unless the results of pre-Cambrian volcanism in the nature of geysers, hot springs, or submarine springs, were extant in the Cretaceous period or later, the pre-Cambrian igneous rocks could not have been the source of the gas. Fig. 2, showing the distribution of volcanic rocks in northeastern New Mexico, gives evidence favoring a causal relationship to the igneous activity of that region. The dikes, the extinct volcanoes and rather widespread distribution of extrusives in northeastern New Mexico certainly suggest, at least, that volcanism may have been an

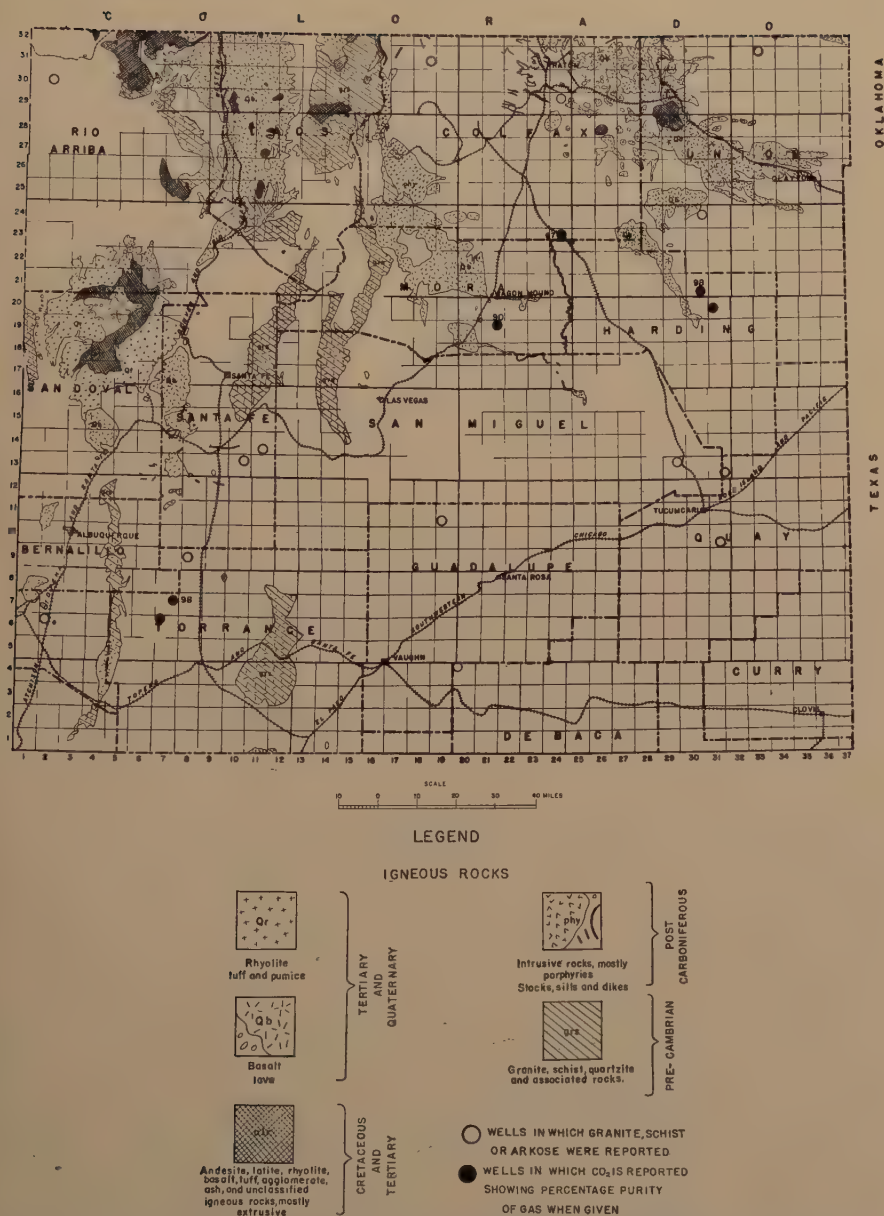


FIG. 2.—GEOLOGIC MAP, NORTHEASTERN NEW MEXICO.

Prepared from U. S. G. S. geologic map of New Mexico, by N. H. Darton and others.

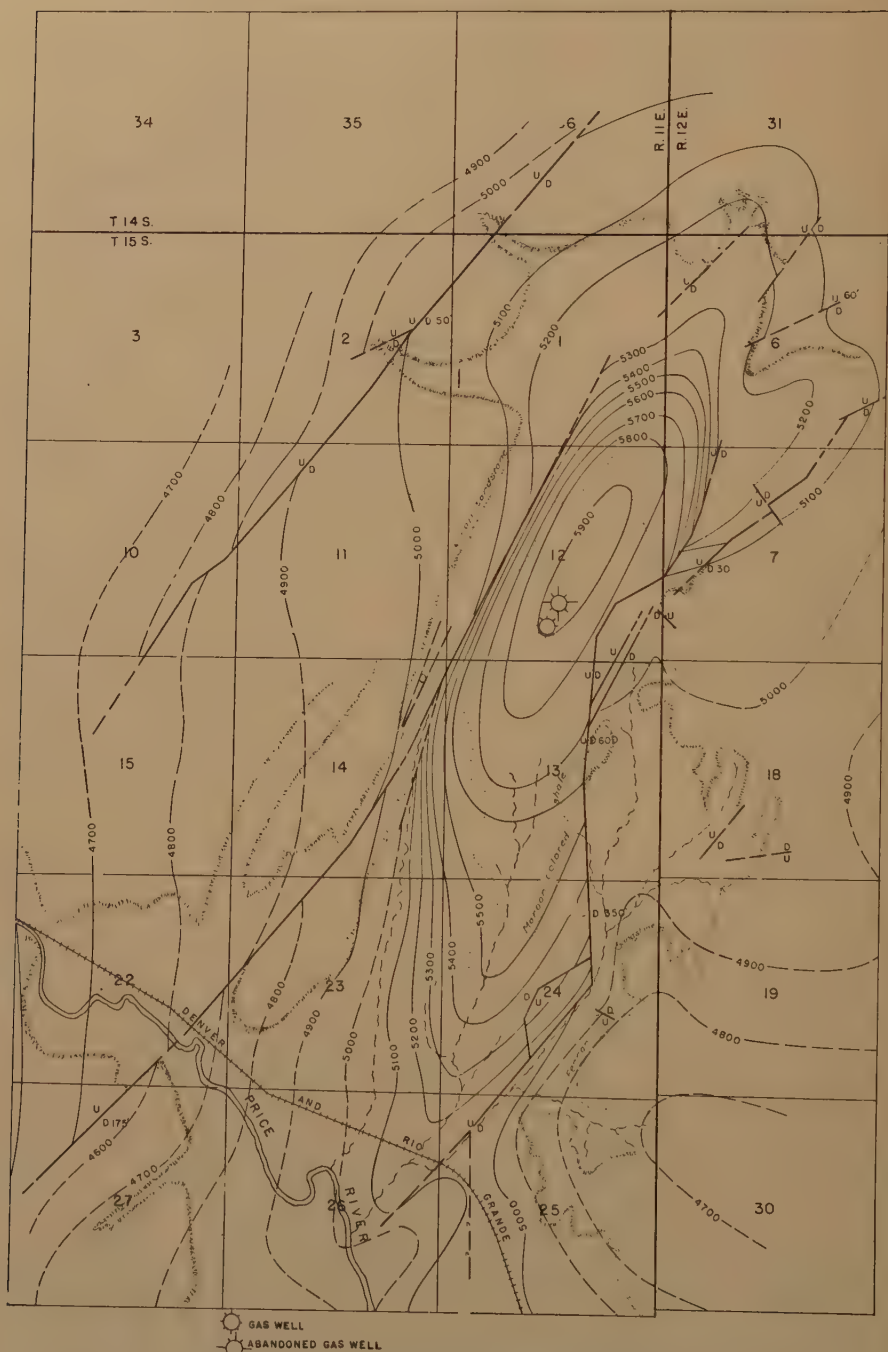


FIG. 3.—STRUCTURE CONTOUR MAP OF FARNHAM ANTICLINE, UTAH.
By J. C. Miller assisted by R. S. Duce.

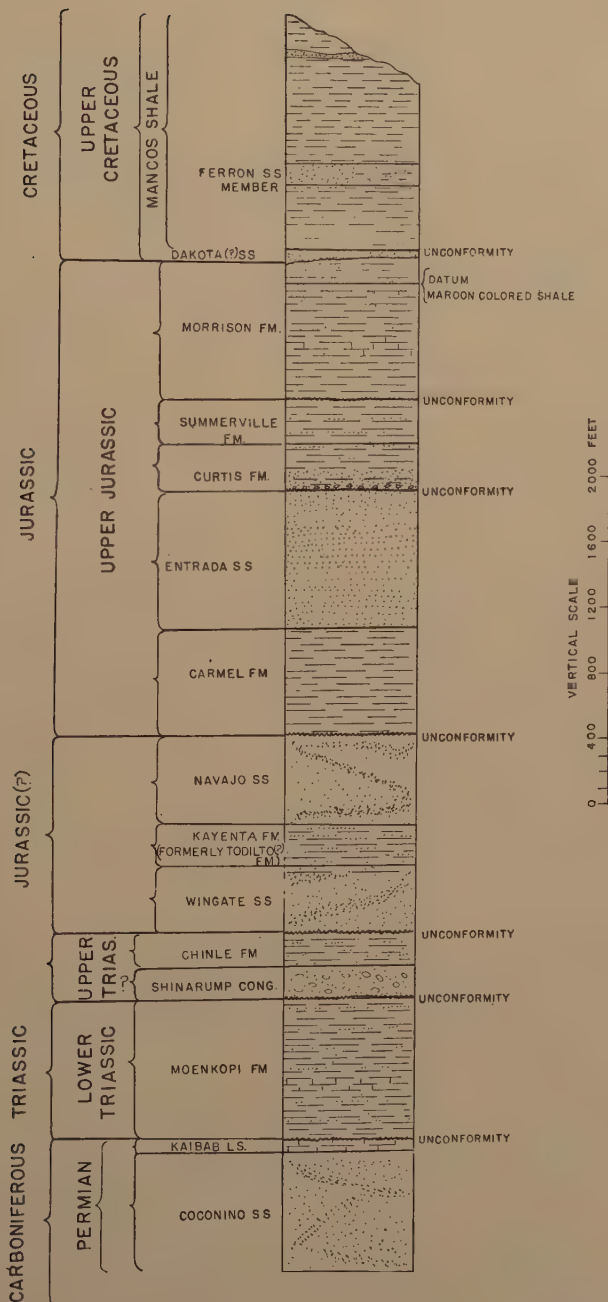


FIG. 3.—(Continued).

Generalized section modified from *Bull.* 806-C by James Gilluly.

important factor in supplying the gas either directly or indirectly in the scattered structures shown as productive of carbon dioxide gas.

PRODUCTIVE GEOLOGIC STRUCTURES

Carbon dioxide in appreciable amounts has been discovered in drilling a number of Rocky Mountain structures in the past but the gas was of no value to the operator or no thought was given to its utilization, therefore many of these wells were abandoned. Of the three areas discussed, the Wagon Mount anticline, New Mexico, is not developed. Both the Farnham anticline, Carbon County, Utah, and the Niland area, California, supply gas for the manufacture of liquid and solid carbon dioxide.

Farnham Anticline.—The Farnham anticline (Fig. 3) is about 140 miles southeast of Salt Lake City, north of the Denver & Rio Grande Western Railroad and the Price River, in Carbon County, Utah. Gas from a single productive well is piped to a 20-ton plant at Wellington, on the railroad, a distance of about 5 miles, where it is purified, liquefied, and converted into solid carbon dioxide, which is shipped to Salt Lake City and the West Coast. The gas analyzed by H. P. Cady, of the University of Kansas, was reported to contain 98 per cent carbon dioxide.

The geology of this area was described by Lupton²⁴ and Clark⁸ and the structure was later mapped by geologists for the Midwest Refining Co. and by others. The first well was started in 1921, by the Utah Oil Refining Co., and was abandoned in 1924 at a depth of approximately 3235 ft. The second well was completed in 1930 by the Carbon Dioxide and Chemical Co. at a depth of 3115 ft. The productive sand is believed to be the upper part of the Coconino sandstone.

The anticline is a small fold on the north flank of the San Rafael swell. Its relation to regional structure has been well shown by Baker,³ in a structure contour map of southeastern Utah.

The structure contours (Fig. 3) are drawn on a rather hard, fissile, maroon or purple shale near the top of the Morrison formation, about 200 ft. below the Dakota (?) sandstone. The Morrison formation crops out along the axis of the structure and sandstone beds believed to be of Dakota age dip steeply, adjacent to two faults on the east and west limbs, which roughly parallel the axis. Except for minor normal faults, the Ferron sandstone member of the Mancos shale forms a fairly continuous rim around the structure. The fault on the steep west flank of the structure appears to be of the scissor type, with a maximum throw near the south end of about 200 ft. The normal fault on the east flank also roughly paralleling the axis has a maximum throw of about 600 ft.

The relative location of the anticline with respect to igneous outcrops is shown on Fig. 4. Carbon dioxide has been encountered in wells in rather widely separated areas in southeastern Utah. No direct evidence was obtained to substantiate a statement that the gas in the Farnham

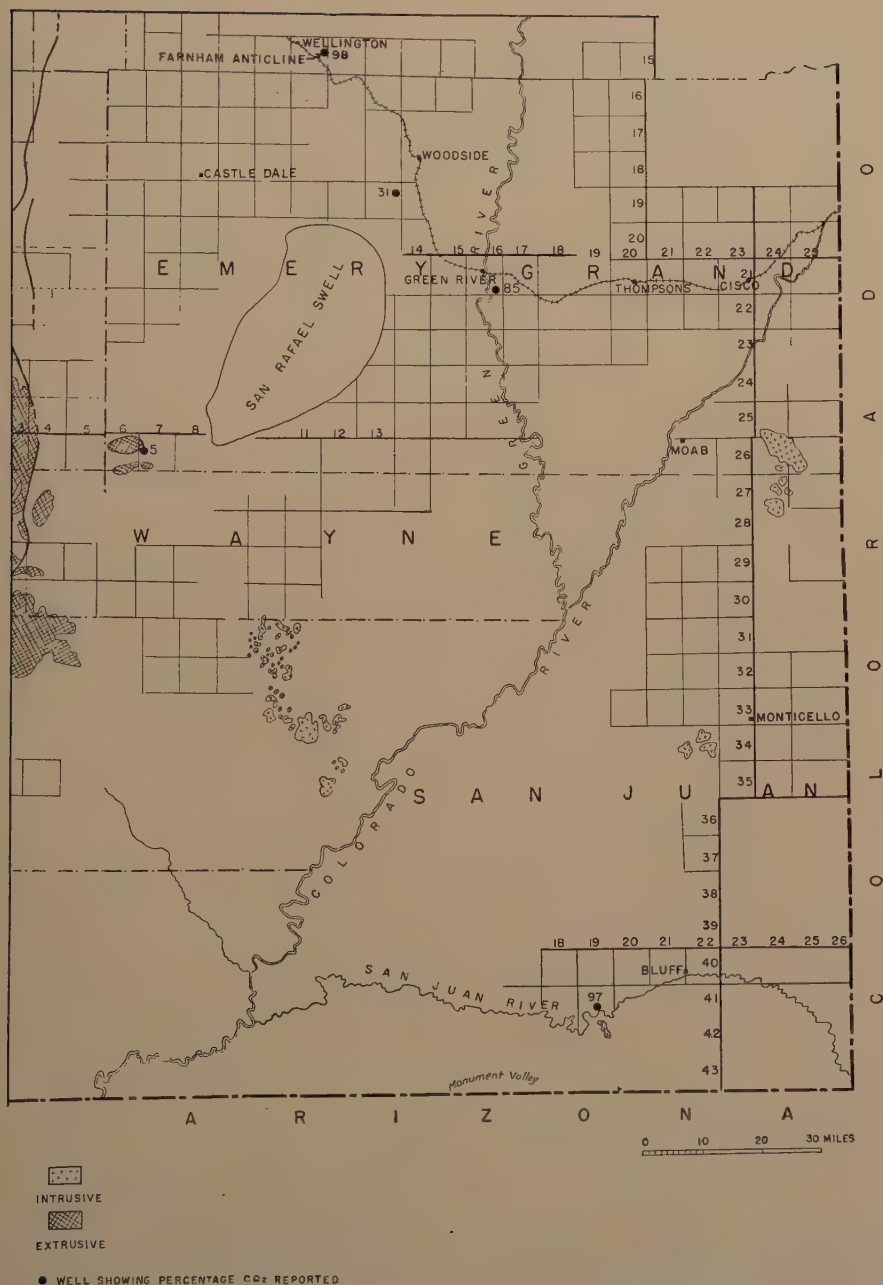


FIG. 4.—DISTRIBUTION OF VOLCANIC ROCKS, SOUTHEASTERN UTAH.
 After reconnaissance geologic map of Utah by B. S. Butler (*Prof. Paper No. 111*).

anticline is of igneous origin; that source merely seems more adequate to account for the relatively large volumes of carbon dioxide, in addition to helium and nitrogen gases found in this portion of the state and adjoining parts of Colorado, by reason of the intrusive and extrusive rocks in and

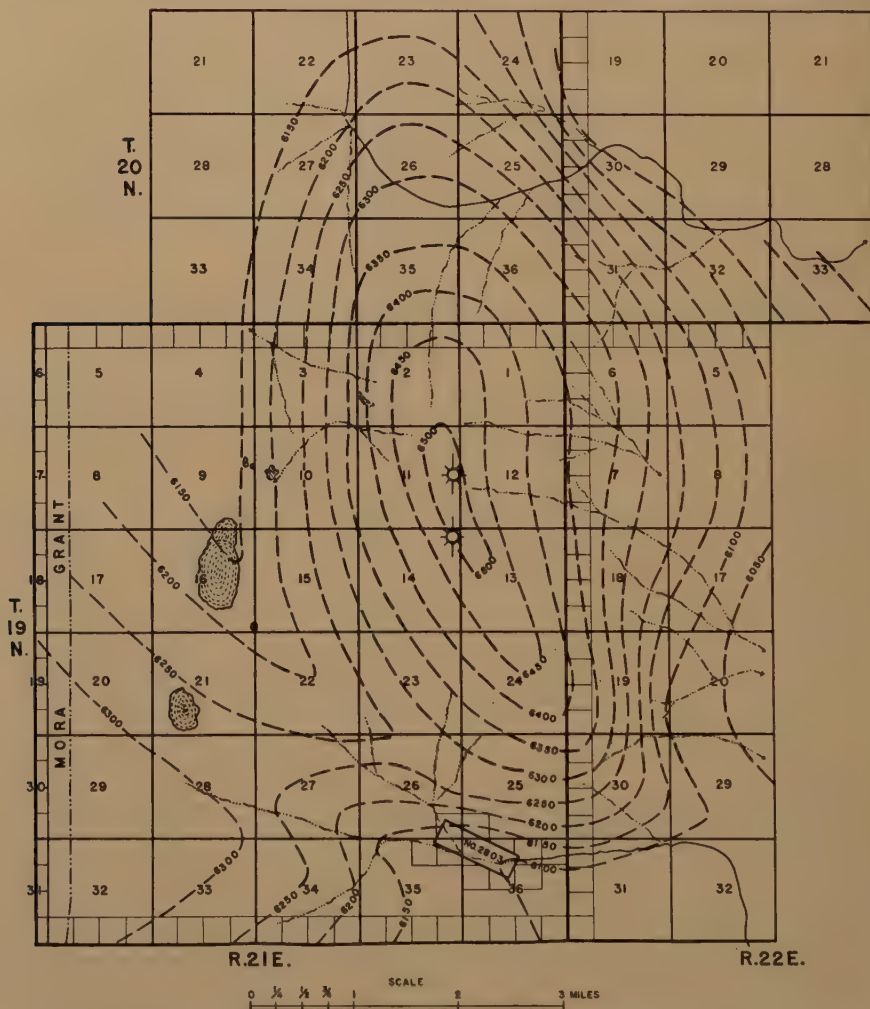


FIG. 5.—STRUCTURE CONTOUR MAP, WAGON MOUND ANTICLINE, NEW MEXICO.
By J. C. Miller and Karl L. Walter.

adjacent to San Rafael swell, also the LaSal, the Henry, and the Abajo Mountains. Numerous dikes are known in the Wasatch Plateau coal fields to the west. South of Price, Wellington, and Farnham, dikes are reported but were not verified by the writer. The strike faults on each side of the anticline might afford means of upward migration of magmatic gases. Nevertheless, the same channels would provide access to the sands or porous formations in the anticline for gases of organic origin.

Wagon Mound Anticline.—The Wagon Mound anticline, in Mora County, New Mexico, is about 8 miles south of the town of Wagon Mound, which is on the main line of the Atchison, Topeka & Santa Fe Railroad, between Raton and Las Vegas.

The first well on the structure was drilled by the Arkansas Fuel Oil Co. during 1925 and 1926 on the SE. $\frac{1}{4}$ NE. $\frac{1}{4}$ of sec. 11, T.19N., R.21E., to a depth of 2613 ft. This well presumably was drilled into the granite after passing through perhaps 400 ft. of arkose, which was logged as hard lime and sand. Non-flammable gas was reported at: 1160 to 1190 ft.; 1420 to 1425 ft.; 1670 to 1675 ft.; 1710 to 1800 ft.; 2220 to 2225 ft.

The upper sand may be tentatively correlated with the Santa Rosa sandstone in the lower part of the Dockum group of Triassic age; the sands between 1420 and 1800 ft. with those of the Manzano group of Permian age, including the Abo sandstone in the lower part; and the sandy lime reported between 2220 and 2225 ft. with the Magdalena group. However, Darton¹¹ considers the record of this well unreliable so that correlation of beds is hazardous.

During 1931 a well was drilled by the Fulton Petroleum Co. in the NE. $\frac{1}{4}$ NE. $\frac{1}{4}$ of sec. 14, T.19N., R.21E., to a depth of 1390 ft. The well is reported to have had an estimated capacity of 600,000 cu. ft. of gas that analyzed 90 per cent carbon dioxide, from a sand of doubtful Permian age between the depths of 1050 and 1135 ft. This gas was reported not to be commercial carbon dioxide gas because of contamination by hydrocarbon gas, and the well was abandoned.

The structure contour map (Fig. 5) resulting from reconnaissance work in 1935 indicates an anticline with the crest trending a little west of north. The Generalized Section shown on this plate was derived from U. S. G. S. Bull. 726-E, p. 176. The structure contours are drawn on a relatively massive sandstone correlated with the Dakota sandstone,

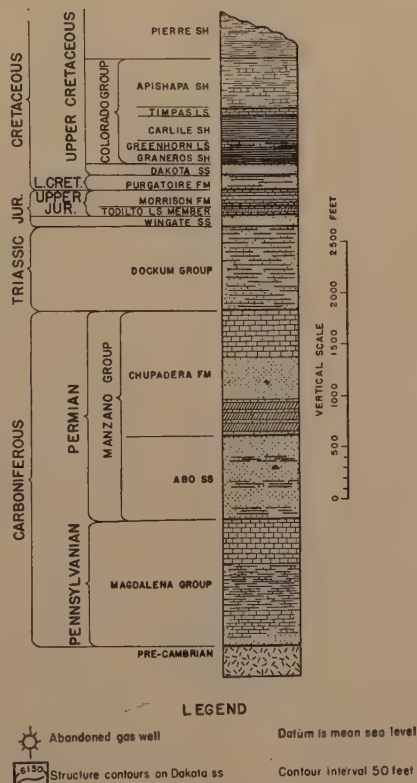


FIG. 5.—(Continued).

Upper Cretaceous, and conform closely to the topography of the area. A remnant of what appeared to be a basalt dike in the SW. $\frac{1}{4}$ of sec. 2, T.19N., R.21E., extending northwest-southeast, was too poorly exposed to verify its extent or its effect on the sandstone beds. However, within a few miles and at the northwest side of the structure, Pliocene (?) and Pleistocene basaltic lavas cap the hills and extend into some of the valleys.

In this area, therefore, there is abundant evidence of igneous activity subsequent to deposition of the sediments, which contain sufficient lime to generate carbon dioxide gas by direct heat. Although the presence of organic source material is not eliminated, the areal map of northeastern New Mexico (Fig. 2) presents abundant visible evidence of volcanism. If this is considered in connection with the discovery of large volumes of carbon dioxide in a number of wells drilled in this part of the state, a conclusion that the gas is of igneous or thermal origin is favored, differing perhaps from the Farnham dome accumulation in that there may have been no extensive lateral migration of gas necessary to cause its concentration in favorable structures.

Niland Area.—The carbon dioxide developments in Imperial Valley, California, are close to the southeast end of Salton Sea, about 4 miles from the town of Niland, on the Southern Pacific Railroad. At present there are two plants in the vicinity manufacturing solid carbon dioxide from gas produced from wells and piped to the plants. This gas is reported 99 per cent pure.

The general geology and structure of the region is discussed by Brown,⁶ who cited some of the early explorations by Williamson, Parke, Le Conte, Blake and others, also later investigations by Sykes, MacDougall, Harder, Fairbanks, Merrill and Mendenhall. Recently a good discussion of the structure was given in a geophysical paper by Soske and Kelly.³⁴

Quaternary deposits immediately underlie the area, except three or four small buttes of obsidian and metamorphic rock protruding from the silts at the edge of the sea. The Quaternary deposits may be regarded as valley fill or lake beds consisting of sand, gravel and clay possibly 1200 ft. thick. The wells find the gas in these beds at depths of 500 to 700 ft. Some of the productive wells are within a mile of the mud volcanoes.

Owing to the character of surface beds, a determination of the structure is not possible except with the aid of geophysics. Soske and Kelly conclude that there is an igneous mass at relatively shallow depth. Its irregular but domelike shape may cause the accumulation of gas and the relatively large volumes of gas are probably primarily due to proximity to the igneous mass, although it is possible that carbon dioxide in the lake beds or similar deposits could be a contributing source at least. The lake beds, however, may act as a seal or impervious layer, preventing upward migration of the gases, which may not be indigenous

with the sedimentary deposits. In view of the results of the Yellowstone Park investigation cited, the writer is inclined to agree with the conclusions of these authors that the gas trapped in the valley sediments of this area is of magmatic origin and that the location and arrangement of the buttes and mud volcanoes have a significant relation to the San Andreas fault, which is believed to extend along the east side of the Salton sink. The Salton Sea basin itself is considered by some writers to be a wedge-shaped graben.

ESTIMATION OF RESERVES

In fields or areas that derive carbon dioxide gas directly from springs, or from igneous sources in general, the quantity of gas ultimately available is indeterminate. Where the gas has accumulated in anticlinal folds or geologic structures that can be mapped with a sufficient degree of accuracy, an estimate of the gas reserves may be attempted. The basis for such estimates, however, involve a number of variable and often unknown factors—for example, temperature of the sand body and areal extent of the gas dissolved in any water present in the sand. There are few data available on reservoir conditions, the rate of depletion or decline, and the amount of recovery in carbon dioxide fields. Therefore, the accompanying figures are to be regarded as an approximation based on available experimental data applied to field conditions, and, in so far as these experiments simulated actual conditions in a gas well, this method of estimation is reliable. It is known that carbon dioxide deviates considerably from the laws of an ideal gas, particularly at very high pressures, and is very soluble in water, two factors of paramount importance in arriving at a fair estimate of future production. On the assumption that 50 per cent of the gas is recoverable when measured at a pressure of 10 oz. above an atmospheric pressure of 14.4 lb. per sq. in., the accompanying calculations are given for a field similar to the Farnham anticline before production started.

The following formula, from U. S. Bureau of Mines publications^{16, 19} modified to suit this special condition, was used in the preparation of the estimate:

$$Q = 43,560 \text{ A}ts \frac{(460 + 60)}{(460 + T_f)} \frac{(P_f d)}{14.4}$$

A = productive area, sq. ft.,

t = thickness of sand, ft.,

s = porosity of sand, expressed decimally,

P_f = absolute pressure in formation, lb. per sq. in.,

T_f = temperature in producing formation, deg. F.,

d = deviation factor at P_f and T_f (in this case the product of $P \times V$ as determined by Amagat)³⁷.

Instead of a pressure base of 14.4 lb. per sq. in., a pressure base of 10 oz. above this figure, or 15.025, was used. It is assumed that free gas exists between the 5700-ft. contour and apex of the structure (Fig. 3) and dissolved gas between the 5500 and 5700-ft. contours, aggregating areas of 25,176,809 and 18,773,489 sq. ft., respectively. A 20-ft. thickness of sand saturated with gas or water of which the effective porosity is 15 per cent is also assumed. The temperature in the sand at a depth of 3100 ft. is calculated to be 95° F. assuming a temperature gradient of 1° in 60 ft. A pressure on the sand or bottom-hole pressure of 882 lb. per sq. in. approximates the well-head pressure 750 lb. per sq. in. plus the weight of the gas. The factor d (taken from Amagat's data for $pv = 1$ at 0° C. and one atmosphere pressure) for 882 lb. per sq. in. and 95° F. is 0.7038, and for the conditions assumed this becomes 0.7967. In International Critical Tables (vol. III, p. 260), the values given for cubic centimeters of gas dissolved by 1 cu. cm. of water at a pressure of 60 atmospheres and temperature of 35° C. are 20.3 and 23.0. Sander³³ gives a value of 22.7 cu. cm. for a pressure of 60 kg. per sq. cm., and to simplify the calculation the higher value was used. Accordingly:

$$Q_1 = \frac{25,176,809 \times 20 \times 882 \times 0.7967 \times 0.15 \times 519.6}{15.025 \times 554.6}$$

$$= 3,309,491,543 \text{ cu. ft.}$$

$$Q_2 = 18,773,489 \times 20 \times 0.15 \times 23 \times \frac{14.696}{15.025} \times \frac{519.6}{554.6}$$

$$= 1,186,559,598 \text{ cu. ft. dissolved or in solution in water}$$

$$\frac{Q_1 + Q_2}{2} = 2,248,025,570 \text{ cu. ft.}$$

if a recovery factor of 50 per cent is assumed, which is intended to include allowance for abandonment pressure in excess of a pressure of 10 oz. above an atmospheric pressure of 14.4 lb. per sq. in. (The atmospheric pressure at the well was calculated to approximate 12 lb. per sq. in.)

The volume of gas Q_2 , absorbed in the water is calculated from the values for solubility of carbon dioxide in fresh water and not salt water. However, it has been determined by McClendon²⁵ and others that:

The change in P_H with change in CO_2 tension is affected only by the temperature . . . Changes in salinity theoretically affect the CO_2 tension and hence CO_2 content of sea water. Fox divides the CO_2 into two parts, one of which obeys Henry's Law and never exceeds 1 per cent of the total. It is this small fraction that is affected by salinity, but the changes in salinity ordinarily met with in the larger oceans change this fraction only about 1.5 percent, and the total change is exceedingly small. We have not been able to determine any effect of salinity on P_H or CO_2 content.

The possibility of diffusion of the gas in contact with the water in the absence of a barrier prohibiting indefinite migration away from points

of concentration has been suggested.* This observation leads to speculation concerning the distance from the apex of a structure that such gas could diffuse in a porous medium before its concentration would be a negligible quantity commercially, eliminating the small fraction that would enter into combination with salts in the water.

If there is no water present in the sand, simply a case of volumetric control as defined by Herold,¹⁵ the volume of gas in the reservoir should be of the magnitude of the free gas volume (Q_1) calculated. If the field is under hydraulic control, the boundary and quantity of dissolved gas on the slopes is continually changing as the free gas is withdrawn and the water encroaches. It is the saturated edgewater that furnishes the additional reserve (Q_2) as this volume of water subsequently advances up the slope. Such advances may be due in part to capillarity but in a sand body of great lateral extent, which probably outcrops on the mountains at a higher altitude, the advance of the water is primarily due to hydrostatic head; in other words, the reservoir is under hydraulic control in accordance with Herold's terminology.

The conditions postulated for this reservoir are therefore a fairly limitless sand body saturated with water, a finite quantity of gas, which may have originated from an igneous source in contact with that sand body, and eventual accumulation of the gas in the anticline.

If it is assumed that the anticline is a minor flexure in a sloping sandy body providing artesian conditions, the hydrostatic head would be greater in the adjacent syncline than on the anticline. If there is movement of water caused by a flow of water from the sand, the conditions may be similar to a long pipe line of which a portion is above the hydraulic gradient, in which event the head at the anticline would be a minus quantity, and there would be a tendency for gas to migrate and accumulate in such areas. It has been common practice, however, to estimate roughly the rock pressure of a gas sand by multiplying the depth by 0.4335; that is, to assume that the pressure is equivalent to the static head of water measured from the surface at that point. This would be about 1344 lb. per sq. in. on the sand at a depth of 3100 ft. in the Farnham anticline well. It is to be noted that a pressure of 882 lb. per sq. in. for the gas in the sand was calculated.

According to this hypothesis the hydrostatic pressure in the sand at the 5500-ft. contour would be in excess of 173 lb. per sq. in. greater than that pressure at the top of the structure. The same result is obtained if the hydrostatic pressure is calculated for the difference in head by reason of a sloping sand or porous strata in accordance with conditions similar to those cited for a pipe line. The apex of the structure, therefore, is a point of lesser hydrostatic pressure even before the

* Oral communication by R. E. Stephens, of the U. S. Geol. Survey.

reservoir is tapped by a well, and there is a tendency of gases and fluids to move toward such areas until the gas is in equilibrium with the hydrostatic head.

In a carbon dioxide gas-water system, diffusion of the gas, which varies with temperature and pressure, is perhaps an appreciable factor, and without a barrier such as a sealed fault or dike or increase of pressure on the edge of the gas reservoir or structure there is little to prevent diffusion of gas throughout the porous medium and its ultimate dissipation. If the temperature gradient is 1° per 60 ft. in depth, an increase of temperature of about 6° and an increase in hydrostatic pressure of 173 lb. at the 5500-ft. contour may result; in other words, an increase in temperature of about 7 per cent, and an increase in hydrostatic pressure of less than 20 per cent. The effect of an increase in temperature is more than compensated by the increase in pressure. In a closed system at approximately equal temperature throughout, the uncombined gas would migrate toward the center of the structure or point of reduced pressure, and not away from the apex; consequently, in addition to the separation effected by gravity, accumulation of gas in anticlines, domes and other suitable traps should follow.

Experimental data show that there is no marked change in viscosity of the gas between pressures of 0 and 60 atmospheres at temperatures between 20° and 40° C.; therefore this factor was omitted from the calculation. The resistance to flow would influence the rate of production and to a minor extent the ultimate recovery but until data are provided by additional experience refinement of the method of calculation seems unwarranted.

In the structure for which the estimate is made, the gas is assumed to be in contact with water, and its withdrawal is assumed to be compensated by evolution of the dissolved gas and encroachment of water until exhaustion of gas is accomplished. If all the water in a sand body of unlimited extent is saturated with the gas the problem is indeterminate and a case similar to that of volcanic springs.

CONCLUSION

The accumulation or concentration of natural carbon dioxide is dependent on one or more of the following factors: (1) proximity to volcanism, extinct or decadent; (2) suitable traps such as anticlines or domes, which are essential to its collection in geologic time; (3) springs or other conduits which provide a means of escape of free or dissolved gas. The recovery of the gas from such deposits may be effected by drilling wells on or near the critical points of these structures, but the number of these wells, whether they be gas wells or water wells, is limited by the rate of movement of the free or dissolved gas through the sand or porous formation and by the volume of available gas derived from the

source rock. The location by a geologist of commercial deposits of this gas is, therefore, not a simple geologic problem and involves appreciation of the probabilities of unsuccessful drilling in areas where surface indications may be otherwise favorable.

ACKNOWLEDGMENTS

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DISCUSSION

(H. I. Smith presiding)

G. S. RICE,* Washington, D. C.—Coming to the subject from an entirely different point of view, I was interested in the origin. There are violent outbursts of carbon dioxide gas in the fields of the south of France and also in lower Silesia, and in both cases I think it is the consensus that the carbon dioxide came from the intrusion of lava into limestone, the beds of which lie below the coal measures. The carbon dioxide liberated by heat worked its way up through fault planes under high pressure and caused these dangerous outbursts in the coal mines. Normally coal gives off hydrocarbon gases only.

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J. C. MILLER.—The question of origin is a controversial one. I agree that the volcanic intrusions acting on lime-containing beds may under favorable conditions produce carbon dioxide. In the MacAllum field, Jackson County, Colorado, there is very little limestone although some of the shales do have a rather high calcium content, but there is no evidence of intrusives. Some writers postulate a high content of carbon dioxide in the sea waters and its migration and concentration in sands. I think the determination of origin, whether organic or inorganic, requires a great deal of research, and I do not know whether it would be definitely answered even then.

P. M. TYLER,* Washington, D. C.—I had some figures in regard to the cost of production of solid carbon dioxide, and I wonder how they compare with making dry ice from other sources. I think most of us here have thought from time to time of the manufacture of dry ice from lime burning or other processes that give off free carbon dioxide. Now usually one thinks of the main cost as being in the raw material. Here there is raw material at low price, but it cannot be as cheap, really, as the carbon dioxide that escapes from the lime kiln.

Is it a matter of the sensible heat in the gas? I presume this is at low temperature and does not have any sensible pressure as it leaves the hole.

J. C. MILLER.—I do not think the temperature of the gas is over 60°, but it depends to a great extent on the average annual temperature. At MacAllum field, the average annual temperature is about 40°. The gas was permitted to expand from about 800 lb. pressure into a 500-bbl. tank, consequently this reduced the temperature to a point where the snow was formed so that the oil came out as a slush.

In the present paper, I broke down some figures that were given me by producers. I could not obtain any figures on cost directly. Two of the plants I visited refused to give figures. I made estimates of the cost of the Diesel installation and from that I was able to obtain a figure very close to the alleged cost of producing the solid. The Bureau of Foreign and Domestic Commerce publishes figures on production and value of output. Cost figures are very difficult to get.

G. S. RICE.—At one time there was a plan to use dry ice for refrigeration of fruit cars coming from California. Has that made progress?

J. C. MILLER.—An attempt to use dry ice in refrigerating cars has been made, but the low temperature that permits use must be properly controlled. Carbon dioxide atmosphere has a preservative effect on fresh fruit and vegetables.

G. S. RICE.—Would not that be a question of adapting the method? Just as you can overheat with your radiator, you may get too great a chilling effect.

J. C. MILLER.—Yes. I think it is interesting that these carbon dioxide wells occur near two of the transcontinental railways. The Southern Pacific, I believe, investigated the possibility of using carbon dioxide in the refrigerating cars but the company has its own water-ice plant, and naturally it means a change of equipment, which might or might not be feasible.

W. L. REMICK, Hazleton, Pa.—Could you tell us something about the system of desiccation? Is it by reduction of temperature, and how much moisture is there in the gas?

J. C. MILLER.—For well gas calcium chloride may be used in the desiccator. There are a number of things that can be used. Usually in visiting a plant information along that line cannot be obtained. Each plant operator is afraid of competitors,

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it seems, and the independents seem to be afraid of the large companies. I could not get the details as to just how much moisture was contained, but the gas was practically moisture-free after treatment. If not, the water would freeze the system.

W. L. REMICK.—The first attempt by one company was to desiccate with sulphuric acid, and the mistake was made of trying to adopt the German system, but to do it under a vacuum. Then it was found possible by circulating and expanding some of the liquid in stages simply to freeze the moisture out of the gas.

J. C. MILLER.—I know that attempts have been made to remove all the moisture before expanding the gas. An effort is made to remove all foreign matter also, because if there is any trace of impurity it ruins the dry ice for use. The chemical and fermentation industries have to contend with the removal esters and other impurities in the byproduct gas.

MEMBER.—In the use of carbon dioxide as an explosive, am I correct in understanding that the ice cannot be desiccated in the same way as the liquid?

J. C. MILLER.—The gas is desiccated before the liquid is formed. My understanding of the manner of its use as an explosive in the Cardox device is that the liquid is injected into the cylinder that contains a heater element.

H. E. CULVER,* Pullman, Wash.—Carbon dioxide is being produced in a small plant in southwestern Washington, in Klickitat County. There have been, I think, a number of freight cars and some five ships, not to mention a number of trucks, fitted for dry ice and used both in local transportation and for transport to the east coast. This ice comes from CO₂ brought up in water wells; it is simply carried above the level to which the water flows, the CO₂ is taken off and run into regular plants. Only about 6000 lb. per day is produced, I believe. That is not a large quantity. At the present time we are trying to find ways and means of locating additional supplies. This gas comes through a very great thickness of basalt and andesite, and apparently is a late emanation from a volcano.

G. S. RICE.—For ordinary refrigeration, has it the same degree of purification as it has for other purposes?

J. C. MILLER.—I think it does. Whether carbon dioxide will be used in a refrigerating system—that is, as much as ammonia—depends on cost of compression and other installations. Refrigerating engineers point out that a carbon dioxide system needs several times the amount of condensing surface that ammonia gas requires.

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Geology, Mining and Processing of Diatomite at Lompoc, Santa Barbara County, California

By HENRY MULRYAN, MEMBER A.I.M.E.*

(San Francisco Meeting, October, 1935; New York Meeting, February, 1936)

THE largest and purest known deposit of diatomite is being actively mined and processed 3½ miles south of Lompoc, Santa Barbara County, Calif., by the Johns-Manville Products Corporation. The workings



FIG. 1.—LOCATION OF DIATOMITE DEPOSIT AT LOMPOC.

cover 4000 acres, the depth of economically recoverable diatomite being 1000 ft. (Fig. 1).

Lompoc, a community of 2500 population, is situated in the Santa Ynez River Valley 10 miles inland from the Pacific Ocean. It is 44 miles

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westerly from Santa Barbara, Calif., and 14 miles north of Point Conception. A branch line of the Southern Pacific Co. connects the city with the main coast route at Surf, Calif., 10 miles west. The branch line continues southward to the deposit, processing plants and warehouses of the company.

HISTORY

The first records of production of diatomite at Lompoc show that the Spanne Brothers, in the late 1880's, removed large blocks of diatomite, hauled them to Lompoc and sawed them with a table saw into rectangular blocks of varying sizes. The blocks were used for the erection of a two-story building having walls nearly one foot thick. In 1892, A. G. and J. F. Balaam hand-quarried lumps, piled them in rows for natural air-drying, and sacked the material for shipment to Captain Barrow at San Francisco. He experimented with the diatomite mixed with tulle



FIG. 2.—FIRST BUILDING ERECTED AT CELITE DEPOSIT, IN 1896.

fiber and produced a fair grade of covering for pipe insulation. The first building, a shed to cover previously air-dried diatomite, was erected by the Balaams at the Balaam quarry in 1896 (Fig. 2).

In 1894 a large block was exhibited at the Midwinter Fair in San Francisco, and aroused considerable comment and interest because of its light weight, color and other physical characteristics. In 1898 one ton of diatomite was shipped to the California Anti-Caloric Co., San Francisco, which was experimenting along insulation lines. In 1902 Harriman and Hanimore came into control of the Dimock mine, near the Balaam property. They hand-quarried, piled and air-dried, and shipped crude material to the California Silica Co. in Los Angeles. In 1904 the Magne Silica Co. was incorporated by Harriman and Hanimore to operate the Dimock mine. They operated under severe financial handicaps for the next four years.

In 1904 crude diatomite from the Balaam mine was being used as boiler and pipe insulation, as furnace lining, in beet-sugar filtration and in the manufacture of a fireproof plaster at the Fireproof Plaster Works, Emeryville, California. In 1906, the Magne Silica Co. erected at Lompoc a simple pulverizing mill for the purpose of reducing the crude lumps to powder.

In 1908 Robert Graham gained control of the operating properties. He was the first person to institute the system of cutting bricks directly from blocks in the quarries. In 1909 the Magne Silica Co. regained the

business, which continued under its management until 1912, when Fitger and Anneke, brewers at Duluth, Minn., purchased the property and changed the name to the Kieselguhr Company of America. They built milling equipment in Lompoc, and later drying and milling equipment at the deposit. Under their ownership a strong effort was made toward the using of diatomite in the filtration of sugar.

During the World War the German word Kieselguhr became unpopular. A new name, Celite, was coined for the diatomite and diatomite products produced at this White Hills deposit. The company was renamed the Celite Products Co., the mine and plant being known as the Celite Company. In 1926 the Celite Company acquired the interests of the Featherstone Company, which was leasing the adjoining acreage on the Hollister Estate, and in 1928 the Celite Company was sold to the Johns-Manville Corporation. The name was changed to the Celite Corporation and later to the Lompoc Factory of the Johns-Manville Products Corporation.*

TOPOGRAPHY

The Santa Ynez mountains form a long, narrow, east-west range bounded on the north by the Santa Ynez River and on the south by the Santa Barbara Channel. They are rounded hills averaging slightly more than 1200 ft. in height. North of the main high ridge are the low, rolling Lompoc Hills, in which the diatomite is contained. These are characterized by numerous canyons, all with exceptionally steep walls but well rounded bottoms. Narrow secondary water courses may be in the center or on either side of a canyon. The hill tops are generally well rounded, or often even flat. The canyons are directly due to structural conditions rather than erosional factors, and usually follow the weakened zones in the vicinity of the axes of anticlines or along fault planes. The high features are invariably in the synclines, where compression forces have caused strengthening of the soft diatomite beds. (See Fig. 3.)

GEOLOGY

The formations exposed in this area are mainly sedimentary rocks, including Cretaceous, Eocene, Miocene, Pliocene and Quaternary. The oldest rocks exposed are serpentines generally assigned to the Franciscan (Jurassic?), but the writer is of the opinion that more detailed work on the exposures at the head of La Salle Canyon (Stewarts Grade) should be done before there is a definite assignment to any given age. Recent

* For further details of early operations the reader is referred to Robert Calvert: *Diatomaceous Earth*. Amer. Chem. Soc. *Monograph* 52 (1930); and V. L. Eardley-Wilmot: *Diatomite. Its Occurrence, Preparation and Uses*. Canada Dept. of Mines, Mines Branch No. 691 (1928).

examinations of San Pasqual Canyon have revealed a thick series of shales and sandstones that may be Eocene or even Cretaceous.

Limestone, limy conglomerate and soft sandstone of Vaqueros age are exposed in San Miguelito Canyon. Lying above conformably are the siliceous shales of the Monterey, attaining a thickness of more than 5000 ft. The hard, thinly laminated shales lie below; the soft diatomite beds, above. The average thickness of the soft beds is 1000 ft. Below



FIG. 3.—AERIAL VIEW OF DIATOMITE DEPOSIT AT LOMPOC.

this are interbedded soft diatomites and opalized hard shales attaining a thickness of slightly less than 1000 ft. Near the top of the soft diatomite beds is a thin bed of conglomerate, which may be considered a transition zone between the upper Miocene and the Fernando. Fernando sands are exposed north, east and west of Lompoc, but not in the area being actively mined for diatomite. Quaternary sediments fill the Valleys.

Pre-Monterey Rocks

Serpentine.—An exposure of serpentine covers about three square miles between La Salle and San Pasqual canyons. It is an igneous rock that shows a fair degree of alteration, which produced a slight mineralization of asbestos. In general, it is not the bluish green shiny rock that usually characterizes California coast range serpentines, but shows less alteration. It is generally assumed to be of Franciscan age, but the writer believes that more detailed work on the exposures and contacts should be done prior to a definite assignment to this age.

Sediments.—Above the serpentine is a considerable thickness of sedimentary rocks variously assigned to the Cretaceous, Eocene, Oligocene and Lower Miocene. Here are sandstones, bituminous shales, carbonaceous shales, limy conglomerates and limestone. These rocks are described by Arnold and Anderson in *Bulletin 322* (Geology and Oil Resources of the Santa Maria District) of the U. S. Geol. Survey (1907).

Monterey Rocks

The Monterey series is of chief interest in this region because it contains the soft diatomite beds. The entire series is slightly more than 5000 ft. thick. The lowest part is composed of hard, siliceous shales, thickly bedded, and showing considerable leaching effect, probably calcareous material having been removed. These beds are badly contorted by sharp folding.

Above these strata are the typical Monterey hard, siliceous, thin-banded shales that are found so abundantly in the coast ranges. They are light brown to chocolate in color, and about 1500 ft. thick.

Conformable to these are interbedded impure but well stratified soft diatomites and hard, glassy, opalized siliceous shales. The diatomite varies in color from white on the outcrop to greenish black when under a thick cover of overburden. The opalized shales are usually black, but translucent in thin fragments. This series is about 1000 ft. thick. Lying directly above are beds of diatomite of varying degrees of purity that are 1000 ft. thick. These are all white on the outcrops.

At the top of the diatomite series are two sandy conglomerates each one foot thick and separated by about 5 ft. of clayey diatomite. The conglomerate marks the top of the diatomite series in the areas under discussion. It consists of well rounded Monterey shales, tuffaceous rocks, and sandstones. Shark bones and teeth are fairly common; shells have not been found.

Fernando

The Fernando sands and sandstones are not important in the diatomite deposit. They are discussed in detail in the bulletin cited before.

Structural Features

The Lompoc Hills contain gently folded synclines and anticlines. The main deposit of diatomite lies in a broad syncline $1\frac{1}{2}$ miles wide and $2\frac{1}{2}$ miles long. Toward the south is a series of small anticlines and synclines that have little of the soft diatomite in them. It has either been eroded away or was never deposited. The more siliceous, hard shales predominate.

Northward is a narrow anticline, a syncline of similar width, and a broad anticline about half as wide as the main syncline. The succeeding

synclines and anticlines contain less thickness of soft beds and show changing conditions of deposition. The beds are usually thinner northward from the main syncline.

All of the present workings are confined to the main syncline, both limbs, the first narrow anticline northward, and the next broad anticline northward. The general strike is N. 65° E. The strike changes to

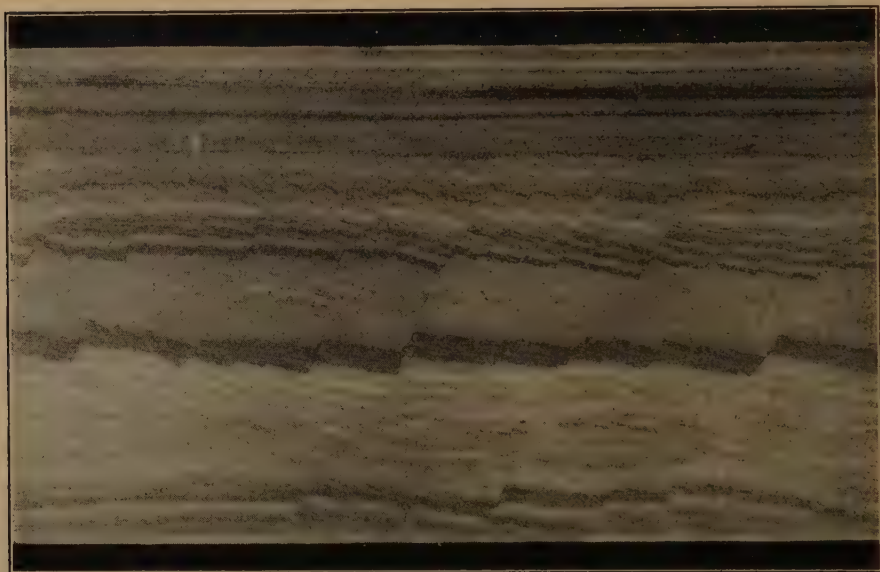


FIG. 4.—NATURAL BRICK SAWED FROM BRICK QUARRY. SHOWS MINUTE FAULTING. NATURAL SIZE.

easterly rapidly near the eastern end. The dips are usually gentle, the maximum being 50° near the bottom of the syncline. As the beds approach the bottom of the syncline the dip flattens to a well rounded basin at the axis.

Faulting

Faulting is unimportant in the main deposit. There is one fault striking nearly west that affects the strike of the beds in the south limb of the syncline only. The displacement has been mostly horizontal with only a slight vertical displacement. Two major faults are found at the northeast and southwest ends of the property. On the southwest end a cross fault has lifted the west block upward, causing the erosion of the soft beds, the only rocks exposed being the formerly underlying hard siliceous shales. On the northeast another cross fault following the Salsipuedes Creek has caused horizontal and some vertical displacement. These two faults coupled with the thinning of the strata northward and

eastward have confined the best beds of usable diatomite to the region controlled by the Johns-Manville Corporation.

One major fault in the second anticline northward roughly parallels the axis, causing a difference in elevation of more than 350 ft. The south side of the canyon is 350 ft. higher than the beds on the north side. What might be termed microscopic faulting has taken place in the diatomite itself. In natural brick are often found many parallel faults showing vertical displacements of less than $\frac{1}{64}$ of an inch (Fig. 4).

As stated before, faulting does not affect the quarrying nor underground workings. It merely serves to confine the best diatomite to the various folds where it can easily be recovered by open quarry or underground mining methods.

DIATOMITE DEPOSIT

The company owns or controls an area comprising 4000 acres in the main deposit. The diatomite contained therein is remarkable because of the high degree of purity in this huge deposit. Its great thickness, more than 1000 ft., and its undisturbed condition, with the added benefit of sufficient relief to permit easy mining conditions, certainly place the deposit in a unique position.

Deposition

How did such a great thickness of diatomite occur? We know that diatoms are microscopic, single-celled, plants having a siliceous skeleton. The marine types prefer clean salt water as their habitat, and thrive best where the water contains sufficient silica for the building up of their highly ornate skeletons.

The upper Miocene is accepted as one of considerable volcanic as well as submarine igneous activity. In the course of such eruptions great quantities of silica—probably as silicic acid in the submarine activity and as slightly soluble siliceous minerals in the land activity—made its way into the ocean.

Interstratified in the diatomites are thin beds of pure volcanic ash whose chemical composition places it as rhyolite in nature. Scattered throughout the entire beds are grains of volcanic ash, which probably have been transported a considerable distance. Here is another source of silica for the diatoms. The beds are remarkably free of material coarser than fine silt. A so-called impure heavy stratum may contain as its only impurity argillaceous material, and that seldom in excess of 10 per cent alumina. Pebbles are occasionally found, but they have always occurred surrounded by excrement, indicating that they were transported there by a bird or fish of some sort. The diatomite surrounding a pebble has a depression at the bottom, and at the top the

stratification conforms to the pebble's shape for probably an inch or less, when it resumes its horizontal plane again.

Some writers refer to the area as probably one of aridity, bays of small extent or an archipelago of small islands.¹

Dr. Frank M. Anderson is quoted by Jordan and Gilbert (p. 14) as follows:

I believe that the diatoms, or the marine species of the Monterey group, are or were northern forms, and lived in the cooler waters of the north Pacific of the Miocene, and were brought south by the ocean currents of the time, and were trapped and impounded in such favorable places as were found along the Pacific Coast of the period. Such favorable places were local and more or less land-locked, or protected areas of sea, free from sedimentation from the land, and where the surface waters of the sea were free to enter and escape, but where they were detained long enough to lose their contents of northern forms under conditions of temperature too high to permit their survival. It appears to me that the deposits of diatomaceous strata, as they are known in California, are all more or less local, and in their occurrence and distribution, conform to this idea.

G. D. Hanna and W. M. Grant found that Miocene diatoms are found as far south as Maria Madre Island, Mexico. They say:²

The diatomite is believed to be Miocene in age. Many of the diatoms belong to species which have previously been found only in Monterey shale of California.

If the diatoms are north Pacific forms, they have covered many degrees of latitude southward, as the Tres Marias Islands lie at Lat. 21 N. off the coast of Mexico due west of San Blas. The diatoms found in the purest beds show little if any abrasion, and have all the indications of having lived in the near vicinity.

Assuming that most of the fish described by Dr. Jordan are shallow-water forms, we can readily subscribe to Dr. Anderson's views that the region was one of protected areas of the seas. However, in view of the absence of detrital material any near-by land masses should have little or no relief. The same results of deposition could be obtained by the diatoms living, propagating, dying, and dropping to the ocean floor in an open sea, facing on a shore line of little relief, such as the east coast of Florida. The same deposition could also take place in sea water far enough away from a land mass of high relief to be free from the effects of erosion. The shallow-water fossil fish would preclude deep water conditions of deposition. It is the author's opinion that the deposit was formed from diatoms that propagated near by in shallow water in protected areas of the sea, where the land had little or no relief.

¹ D. S. Jordan and J. Z. Gilbert: Fossil Fishes of the (Miocene) Monterey Formations of Southern California, 13. Stanford University Publications (1919).

² G. D. Hanna and W. M. Grant: Expedition to the Revillagigedo Islands, Mexico in 1925, II. *Proc. California Acad. Sci.* (1926) 15, No. 2, 116.

Effect of Seasonal Rainfall

An interesting feature of the thinly stratified beds is the succession of extremely thin, nearly pure strata, and likewise thin but slightly "clayey" strata. In all thinly laminated diatomites these strata are noticeable at once by the slightly darker color of the "clayey" strata over the nearly pure diatom strata. The effects of rainfall erosion, amount of microscopic silt in the water at various seasons, all can be seen in a given specimen.

On a larger scale we can take a series of the thinly laminated beds, say 20 ft. thick, underlain by one 4 ft. thick that is only poorly laminated, darker colored, of which the particle size is very much greater, and which is definitely higher in clay or silt content. Underlying this may be a series of 40 ft. or more of poorly stratified material, which usually ends with what is called a "solid clay" band 1 to 6 in. thick. Immediately below are found more of the thinly laminated beds.

The relationships of these pure and impure beds as well as those of the tiny interlaminae with seasonal rainfall and erosion is a problem worthy of further study.

Diatomite

The diatomite in the 1000-ft. section can be classed as three kinds: (1) pure thinly stratified, (2) impure conchoidally fracturing, and (3) impure clayey coarsely stratified. These are in their order of economic importance. To understand properly their stratigraphic relationships it is necessary to mention that the thin conglomerate and coarse sand conglomerate are at the top. Below this lies the impure unstratified conchoidally fracturing diatomite. Under this bed are alternating pure thinly stratified beds and the impure clayey but coarsely stratified beds.

Thinly Stratified Beds.—The most important beds economically are those that are thinly stratified. This diatomite is white, soft, porous, and strong. A natural brick $2\frac{1}{2}$ in. thick, $4\frac{1}{2}$ in. wide and 9 in. long will stand pressures of over 100 lb. per sq. in. when applied at right angles to the stratification. Stratification lines vary in thickness from a hair line to $\frac{1}{4}$ in., the average being about $\frac{1}{32}$ in. Some are nearly microscopic in thickness.

The diatomite in the bank is always slightly gray or even light brown, but when dried thoroughly it becomes almost white. The presence of moisture makes coloring matter more noticeable. On the surface the diatomite is always lighter in color than underground. The difference in color in the upper zones is due to oxidation and removal by leaching of the impurities and extremely fine diatomaceous material.

The diatomite in place is always moist. The more porous beds may contain a minimum of 20 per cent water; the less porous, 30 per cent at

the surface. A few feet below the surface these percentages increase rapidly to more than 50 per cent at the main tunnel level, which is well above the free-water level. When saturated the diatomite contains slightly less than 75 per cent moisture. It is interesting that during winter months the surface rocks contain more moisture than in summer, but that shortly below the surface the diatomite carries a constant percentage.

Capillary attraction plays an important part in this constant moisture content, and also explains the presence of efflorescent soluble salts on some outcrops of diatomite. Continual drawing up of water from below also brings soluble salts to the surface where evaporation takes place,



FIG. 5.—COSCINODISCUS. $\times 700$, REDUCED $\frac{1}{3}$ IN REPRODUCTION.

with consequent deposition of the salts at the surface. This is particularly true on the outcrops of less pure beds of diatomite. The zone where this evaporation and concentration of salts is effected is very narrow, only a few feet deep. Some efflorescences occur in tunnels that have been opened for long periods of time; the concentration being greatest on the least pure beds. Efflorescences always occur in underground workings that expose greenish diatomites.

The thinly stratified beds to the 500-ft. level can be considered as white in color. They are the purest beds, not only in silica content, but also in diatom structure. Chemical compounds other than organic are few, and make up only a minor amount of the total.

Celite is composed principally of a mixture of different sizes of filiform or needle-shaped diatoms, disk-shaped diatoms and fragments of each. The filiform, or spicular, diatoms are mainly *Synedra*, *Thalassiothrix*,

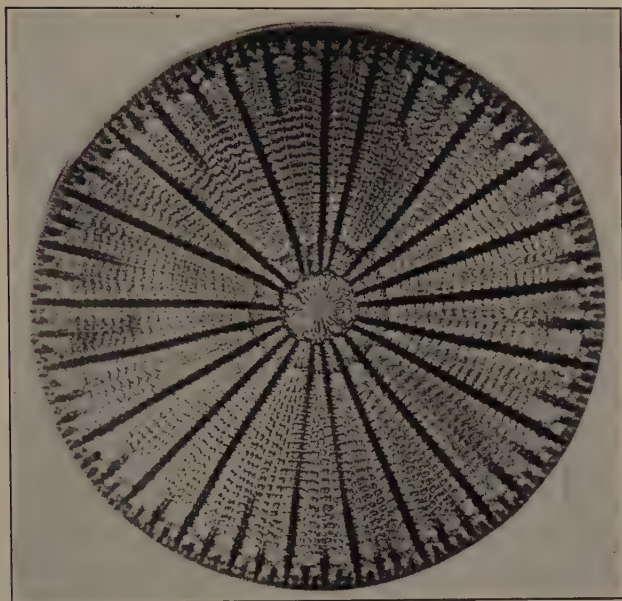


FIG. 6.—ARACHNOIDISCUS. $\times 720$, REDUCED $\frac{1}{3}$ IN REPRODUCTION.

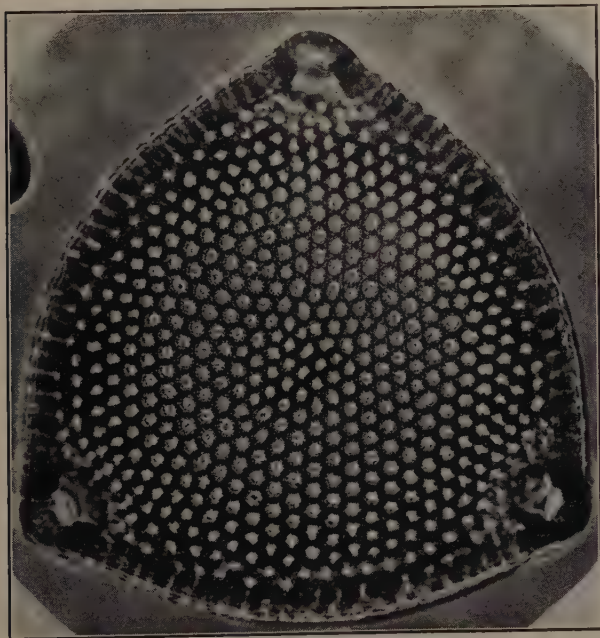


FIG. 7.—BIDDULPHIA. $\times 1000$.

Thalassionema and *Nitzschia*. The disc diatoms are principally *Coscinodiscus*, *Arachnoidiscus*, *Actinopterychus*, *Aulacodiscus*, *Auliscus* and *Asterolampra*, of which *Coscinodiscus* is the more prevalent discoid form (Figs. 5 and 6). The percentage of filiform to disc shapes varies considerably throughout the deposit. Some strata contain as little as 15 to 20 per cent of filiform shapes whereas other strata may run as high as 75 or even 85 per cent filiform types.

Over 200 varieties of other diatoms have been noted as accessory forms in Celite. The organisms other than diatoms are silico-flagellates, radiolaria and sponge spicules. No foraminifera have been observed in

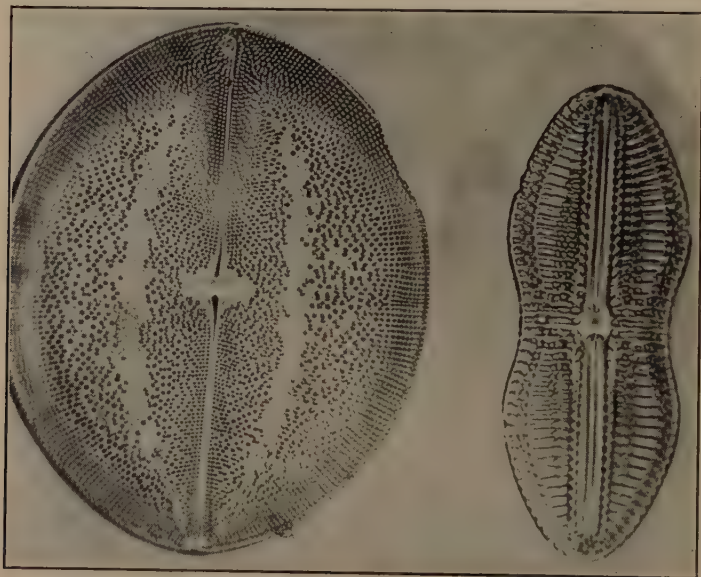


FIG. 8.—NAVICULA. $\times 750$.

the deposit. Some of the accessory forms of diatoms include *Navicula*, *Biddulphia*, *Dicladia*, *Xanthiopyxis* and *Rauxia* (Figs. 7 and 8).

The principal impurities of Celite are very fine quartz grains, clay particles, amorphous silica, minute flakes of volcanic ash, traces of inorganic salts and small amounts of organic matter.

Impure Conchoidally Fracturing Beds.—The second variety of diatomite is also light-weight, white to cream colored, but shows fewer of the minute stratification planes. Generally it has a conchoidal fracture. Changing depositional conditions produced this type of diatomite.

Stratified "Heavy" Beds.—These are the heavier argillaceous diatomites that are interstratified with the pure, thinly laminated ones. They represent periods where the sea water became dirty with clay, silt and volcanic ash. Diatom deposition continued, but not nearly so rapidly. The beds contain as much as 10 per cent alumina. The diatomite in the

least stratified portions is slightly plastic when well wetted. It is notable that even the most impure diatomites have an apparent density of less than 1. All of them will float on water an appreciable period before becoming saturated and sinking. Even in these "heavy" beds the cleavage is roughly horizontal.

Volcanic Ash

Much has been written about Miocene volcanic activity and the presence of tuffs and ash in the Monterey shales. Here the volcanic ash appears in two ways; scattered throughout the beds of diatomite or as interstratified streaks. The volcanic ash is extremely fine in grain, seldom larger in particle size than the diatoms themselves. The grains are always platy, light colored, and have sharp angular outlines. This sharpness of outline indicates little change by erosion, and also that they were deposited as dust in the sea water very close to the point of deposition.

The more important occurrences are the streaks that are found at definite horizons. They may vary in thickness from $\frac{1}{16}$ in. to as much as 6 in. The color is always a bright silver gray. Chemical analyses place them in the acid class of volcanics, quite possibly rhyolite. Invariably, these streaks contain few if any diatoms except at the tops and bottoms. The centers are nearly pure volcanic ash of a nearly uniform particle size.

In one instance two pure diatomite strata are divided in the middle by volcanic ash having a thickness of not more than 2 in. This stratum of ash is present everywhere on the property, and always at exactly the same horizon. This is true of three other streaks $\frac{1}{2}$ in. wide and of the one 6 in. wide that occurs low in the structure.

"Flints" in the Soft Diatomites

Hard, opalized shales, locally called "flints," are interstratified with the soft diatomites. The author, knowing the great diversity of opinion about them, will not attempt to establish the cause of their origin, but will simply state the facts regarding their occurrences in this deposit.

There are several kinds of "flints": (1) hard, nodular, concentrically banded; (2) the same as type 1 but showing a high degree of leaching; (3) interstratified, black, opalized strata showing exactly the same banding as the enclosing soft diatomites; (4) hard, white, intermittent calcareous flints; (5) opalized flints that show gradation from hard glass to extremely soft, porous, spicular diatomite.

All "flints" except Nos. 4 and 5 are located at definite horizons throughout the property. They are always present in the same strata wherever they occur, in synclines or anticlines, at the top of a structure or at the bottom. They always occur surrounded by diatomite of

extreme purity. They seldom occur in the less pure beds. Flints 4 and 5 may appear anywhere in the diatomites.

General Uniformity of Diatomites

The diatomite in the main synclinal structure is remarkable for its degree of uniformity wherever it has been examined. The chemical analysis of a given stratum will vary only slightly from top to bottom, except for the change where hydrocarbons and organic material are found. The diatom content, that is, the mixtures of certain types, is practically constant. A particular stratum will show little if any change

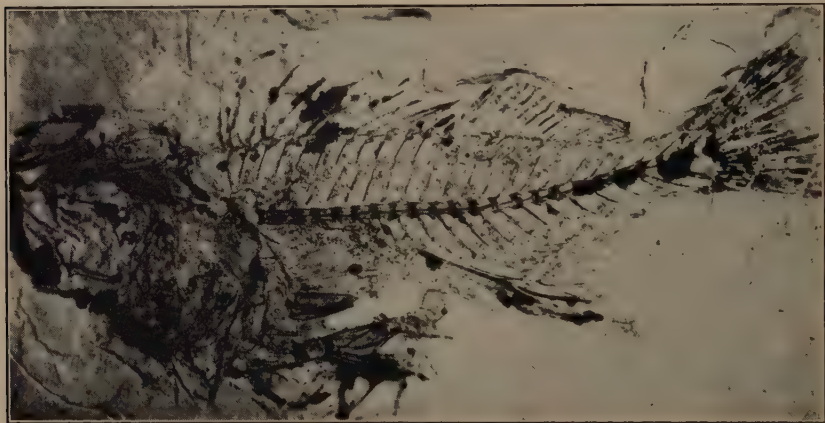


FIG. 9.—FOSSIL FISH OBTAINED RECENTLY.

at any level or in any quarry. Regular horizons are established by thin clayey streaks, certain flints, volcanic-ash streaks, and many other tiny features that ordinarily would not be noticed.

A 6-in. solid, hard, glassy flint showing rhythmic banding is found everywhere 25 ft. above this $\frac{1}{32}$ -in. seam of dark material. The 2-in. volcanic-ash seam has already been mentioned. Such regularity of beds of pure material and impurities certainly indicates a quiet depositional period in a region little affected by erosional or other agencies.

Fossils

The author has found no foraminifera. Microscopic life other than diatoms appears to be composed of large quantities of sponges (usually of a needlelike or spicular nature), silico-flagellates, and occasional radiolarians. No fossil shells have been found, other than two rock borers found in a piece of opalized shale 6 in. square by 3 in. thick. Numerous concentric nodules, seldom more than 1 in. in diameter and $\frac{1}{2}$ in. thick appear in various strata. These are green in color, soft, and are generally believed to be the remains of soft organisms.

Fossil fish are numerous. Dr. Jordan describes some of them in the publication quoted. In general, they occur singly with no relation to each other or to any given stratum. (Fig. 9.) One outstanding fossil is of one of a school of *Xyne Grex* (Fig. 10), an extinct type of herring, according to Dr. Jordan. These fish, uniformly 6 to 8 in. long, lie in a pure diatomite stratum 6 ft. below the 2 in. volcanic ash locally called "silver sand." They are packed solid, but seldom overlap each other. They occur 1000 ft. along the strike and 250 ft. down the dip directly behind the milling buildings. Every part of the fish is preserved, except,

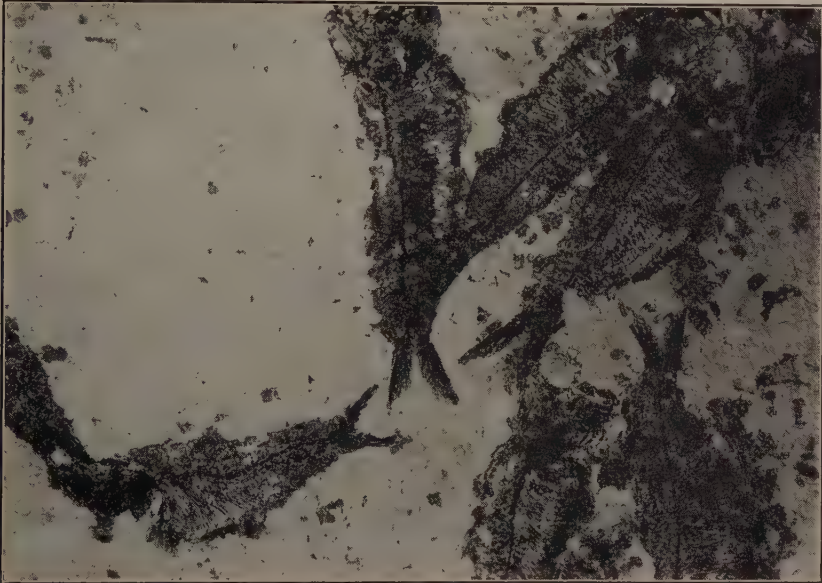


FIG. 10.—FOSSIL *Xyne Grex* FROM FISH STRATA.

of course, the flesh. The tails, fins, backbones, ribs and scales are always in excellent condition. The head does not preserve so well but it is always well defined. Sponge spicules are usually well concentrated around the fish remains. The single fishes, often found, vary in length from $1\frac{1}{2}$ in. to more than 3 ft. Circles of sponges always are found with them.

The cause of the death of the fish in such great numbers is one of conjecture. Possibly it was due to a change in alkalinity of the sea water, temperature variations, or the presence of poisonous solutions probably emanating from submarine volcanic activity near by.

Fossil birds are found occasionally. They are fairly well preserved specimens of shore or wading birds³. One piece of silicified drift wood

³ An excellent description of a few fossil birds is given by Loye Miller: Avian Remains from the Miocene of Lompoc, California. Carnegie Inst. of Washington Pub. No. 349, 107-117.

was dug out of a pure diatomite bed. It had become embedded in the diatom ooze, its woody structure being replaced by silica, all of the original structure, including the bark, being excellently preserved. Pebbles are so rare as to be curiosities.

Larger marine life consists of the skeletal remains of seals, sea lions, sea cows and whales. One backbone of what is believed to be a sea lion reposes in a museum case in the plant office. Numerous whales have been found, especially in the area where natural brick is sawed. One toothed whale was removed, the teeth being well polished and rounded.

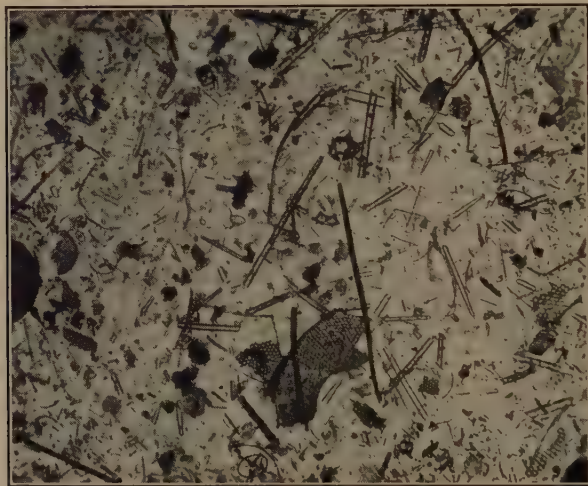


FIG. 11.—STREWN SLIDE SHOWING TYPE OF MATERIAL CHARACTERISTIC OF CRUDE DIATOMITE. $\times 160$.

The length of the longest tooth was $3\frac{1}{2}$ in. In general, the inside of a whale's head is always replaced by a hard mixture of silica and calcite.

At the very top of the formation is the conglomerate containing innumerable bones and occasional shark teeth, the latter ranging in size from $\frac{1}{4}$ to 3 in. in length.

Microscopic Examination

As the quality of diatomite and its utilization are due to the numbers, types and mixtures of diatoms present, it is necessary to check the testing microscopically. The powder is weighed out, distilled water is added and enough of the agitated liquid and powder is withdrawn to cover a cover glass. These covers are dessicator dried, mounted in Hyrax, and examined with an ordinary microscope. The outlines of the diatoms are very sharp in Hyrax (Fig. 11).

A predominance of fines, broken diatoms, argillaceous or volcanic-ash particles, indicates a diatomite of little value for filtration uses. An even mixture of discs and needlelike forms usually indicates a high quality of

filtration crude. In like manner, diatomites for admixtures, fillers and insulation can be determined microscopically.

MINING METHODS

The original mining methods consisted of hand removal of diatomite where there was little or no overburden. It was split into lumps, piled in rows 6 ft. high and 3 ft. wide and allowed to dry in the flat fields receiving the greatest amount of wind and sun. In 45 to 60 days (except in winter) the diatomite had lost 30 to 40 per cent in free moisture, there being usually not more than 8 to 10 per cent remaining. This method was necessarily confined to areas where the overburden could be removed with little effort and cost, Fresno scrapers pulled by mules being used.



FIG. 12.—DIESEL-POWERED SHOVEL LOADING CRUDE DIATOMITE INTO 14-YARD DUMP TRUCK.

The work was slow and expensive, the loss in the form of fines as high as 50 per cent.

In time, improved methods were developed. Double-drum or tandem-drum electric hoists pulling locally designed toothed buckets scraped the overburden to the edges of the quarries. Channeling saws were developed to saw the diatomite into 4-ft. cubes. A bottom cut was made on a 25° dipping face, and the blocks were loosened with picks. The blocks slid to the quarry heel, where they were split into large lumps, loaded to wagons, hauled to the drying fields and piled for air-drying. It was found that some diatomite could also be quarried directly with draglines. The buckets emptied into chutes, which unloaded to bottom-dump wagons, and later to automobile trucks.

Gasoline-shovel and surface train loading came later. The trains have become obsolete as well as aerial tramming, and now gasoline and Diesel-powered shovels with trucks perform all overburden stripping and diatomite quarrying (Fig. 12).

All transportation of diatomite to the primary crusher is by electric trolley trains underground.

Surface Quarrying

The beds of diatomite vary in dip from flat to 50°. The overburden ratio becomes so high that it is possible to quarry on only one side of a fold at a time. It is not possible to work circular open pits, owing to the dip of the strata and the requirement of selective quarrying.

The layout of the overburden removal and diatomite quarrying is generally the same for each quarry, except for local variations. The general trend is toward excavation by power shovel and transportation by trucks. Roads are graded, graveled and smoothed for rapid movement to the dumps. Quarrying by shovel and trucks begins at the outcrop, working along the strike to the end of the quarry, and then making new cuts at successive intervals down the dip.

Drilling and Blasting.—Drilling and blasting are seldom done. The shovels are powerful enough to quarry the diatomite without aid. Blasting is done only where it will speed up the operations or in the removal of thick layers of "flint."

In drilling down holes in diatomite, compressed-air jackhammers are used. No water is added, the cuttings being blown out. As they contain 40 to 50 per cent water, there is no dusting. They are usually sprung with 40 per cent dynamite and No. 8 electric caps, the holes later being filled with granulated powder and detonated with No. 8 electric caps. A hand battery or 110-volt electric line is used for firing. When drilling flints the jackhammer is used to drill holes seldom more than 3 ft. deep. A series of holes is loaded with 40 per cent dynamite and electrically detonated. If more than one parallel series is to be blasted, delay electric caps are used.

Dragline Excavation.—Some quarries can be more economically worked with dragline buckets.

A permanent set-up is made, power line established, and the bucket proceeds to remove overburden to the quarry edges; or, in diatomite recovery, drags the material into vertical storage shafts. This method is slower than shovel work, but is quite satisfactory in some cases.

Underground

Glory Holes.—The primary crushing plant being located underground, receives its diatomite through vertical shafts, known locally as "glory holes," and the main transportation tunnel. The holes are all circular, varying in diameter from 12 to 36 ft. They are unsupported inside, the walls showing very little tendency to cave even when the holes are empty for some time. All holes have concrete chutes at the top to prevent wearing away of the walls when diatomite is dumped into them. The shovels load diatomite to the trucks, which haul it to the glory holes,

where it is unloaded (Fig. 13). The underground train hauls it away at the bottom as required by the milling plant.

Tunnels.—The total footage of actively used tunnels is 9614 ft., all of which is contained in the main transportation system. The main tunnel, 7000 ft. long, has branches running to various glory holes at selected locations. Little timbering is required. Only in zones where the roof is weak are caps and posts used, the caps being niched into the walls. The stations where the tunnels branch are all timbered with



FIG. 13.—LOADING UNDERGROUND FROM VERTICAL STORAGE SHAFT (GLORY HOLE).

All columns and supports for glory-hole bottom, except one at extreme left, are concrete.

treated lumber and loosely lagged on the roof. The glory-hole bottoms are concrete, some of the posts being precast.

The driving of a tunnel is comparatively simple, as the rock drills and breaks well, grades are maintained easily, the ventilation is good, and there is no dust or water with which to contend.

Ventilation.—In the driving of the tunnels and raising of the stopes the smoke and gases are removed by blowers until a raise on the dip can be put through to the surface. These are run through at regular intervals and provide excellent natural draft.

The moisture being as high as 50 per cent makes mining operations dustless. There is no free water in the present underground workings.

Timbering.—Where timbering is required, in stations having openings too wide for safety, treated 8 by 8-in. or 10 by 10-in. timbers are placed as for a drift set and lagged loosely on the top with 2 by 12-in. rough boards.

It becomes necessary in weak zones to catch up the roof occasionally, and for this posts and caps are used, the latter being easily niched into the diatomite walls.

The bottoms of glory holes used to be supported by timber, but the new trend is to make them of concrete with concrete posts for supports. In some areas, especially in the vicinity of large glory holes, concrete precast caps and posts are used for ground support.

Transportation.—All underground transportation is by means of trains powered by electric trolley locomotives.

PRIMARY CRUSHING PLANT

All diatomite destined for pulverizing into powdered form passes through the primary crushing plant. The electric locomotive hauls the trains of diatomite underground from the storage shafts (glory holes) to the crude bins. It is then fed by hand to a belt conveyor underneath, which conveys the material to a short cross conveyor that in turn carries it to the milling machinery.

A second bin having an automatic unloading bottom feeds directly on to the cross belt having a magnetic pulley at its front end. Tramp iron is thus bypassed from the milling system. Lumps of diatomite not more than 30 in. wide and 1 ft. thick are fed over the magnetic pulley into a pair of revolving rolls set with 4-in. spikes.

The coarsely crushed material drops by gravity into one of two high-speed mills. The grid openings on the bottom are changeable at will. The largest particles are 1 in. square by $\frac{1}{2}$ in. thick, the average being less than $\frac{1}{4}$ mesh.

The primary crushing plant has capacity sufficient to take care of all powder requirements daily.

The bar-milled diatomite is belt-conveyed to any of six storage bins, which feed directly into the secondary milling systems.

Under normal operating conditions the train crew loads the cars, hauls them to the storage bins, and then mills the diatomite. This is a three-shift operation.

NATURAL BRICK

The method of recovering standard size insulating brick is unique at Lompoc. Blocks of natural diatomite are sawed by machines in the quarries to standard dimensions of $2\frac{1}{2}$ by $4\frac{1}{2}$ by 9 in. A quarry suitable for the sawing of brick is stripped of overburden by shovel or dragline excavation, the floor smoothed to the dip of the stratum. To the present the dip has not exceeded 25° on these quarries.

A heavy steel track is laid across the quarry face parallel to the strike (right angles to the dip). Upon these rails are mounted three machine saws operating electrically and hydraulically. The machine in the center is a face cutter with circular saws and inserted teeth turning at right angles to the strike, which saw the lengths of nine bricks. The machine head cuts rapidly downward in a direction at right angles to the floor for 5 ft., and into the bank for $4\frac{1}{2}$ in. The machine is moved to

the end of the last channel and the head cuts rapidly, upward this time, to the top of the bank.

In the first cut the lengths have been sawed $4\frac{1}{2}$ in. thick. A similar number of bricks were cut to length on the upward cut. This process continues until the bank $4\frac{1}{2}$ in. deep has been channeled for the lengths of all the bricks on the 5-ft. bank. The next operation is to saw the widths and thicknesses. The machines on either end have two sets of saws mounted on their heads. The saws are circular and have the same type of inserted teeth. One motor has three circular saws mounted $2\frac{1}{2}$ in. apart, the saws revolving in a plane parallel to the stratification. The other motor has one circular (tail) saw mounted behind the one just mentioned, the rotation of which is parallel to the 5-ft. bank (right angles to the three saws). It is set in far enough to saw a $4\frac{1}{2}$ -in. cut. A machine moving continuously across the quarry floor then has three saws whirling against the bank and sawing the thicknesses of two bricks, while another saw, the tail saw, whirls parallel with the bank, sawing out the $4\frac{1}{2}$ -in. widths of the two bricks. The lengths have been sawed previously by the center machine, the thicknesses and widths sawed nearly simultaneously by one of the other two saws or by both operating in opposite directions.

When the saws have reached the ends of the quarry floor the machine direction is changed, saws with new or sharpened teeth are attached and the journey is resumed back to the original starting place, sawing the widths and thicknesses of the next two lower bricks. This operation continues until 18 to 20 layers $2\frac{1}{2}$ in. thick have been sawed out, leaving a smooth face again. The face cutter, sawing vertically, is brought into use and the entire bank again channeled for the lengths of the bricks, whereupon the two moving machines resume sawing the thicknesses and widths. The length of the tubes of the face cutter allows sawing for 5 ft. down the dip. When this limit is reached the track with the saws is lowered down the hill 5 ft., aligned and braced, and sawing operations are resumed as before.

As the tail saw cuts the widths the bricks are loosened from the bank and lie on the solid bank below. They are then sorted and placed in boxes. When a box is full, it is attached to a dragline, which hauls it to a point under one of two aerial conveyors known as Ruggles hoists. The boxes are picked up by the Ruggles hoists, conveyed by aerial cable to a vertical shaft into a tunnel and landed on flat cars. A train load powered by a Ford-engine locomotive is hauled to the brick plant for further handling.

Drying

The bricks are transferred by hand from the quarry boxes to tunnel drier cars at the brick plant (Fig. 14). A second inspection takes place here and bricks with broken corners, split bricks, etc., are thrown out.



FIG. 14.—BRICK PLANT AND WAREHOUSES IN FOREGROUND. MILLING PLANT, WAREHOUSE, MACHINE SHOP AND STORE-ROOM IN BACKGROUND.

The drier cars are forced by a hydraulic ram through tunnel driers at a regulated rate at temperatures up to 500° F. After thorough drying the cars of bricks are transferred by trolley car to the sizing machines.

Sizing and Packing

In the quarries it is easy to control the lengths of the bricks, but the dimensions of widths and thicknesses show variations due to uneven floors, worn teeth and many natural causes. Sizing machines are employed to provide uniformity in these dimensions.

By a cross conveyor belt the bricks are carried to a belt traveling at right angles to it. They fall upon this belt with their lengths toward the sizing wheels (carborundum) and the wide sides up. The bricks pass between two wheels and the 4½-in. width is smoothed off. Passing on they turn so that the 2½-in. side is upward and then pass between two more sizing wheels where that dimension is smoothed. The finished bricks pass to a steel plate, where an inspector gives them a final examination. Bricks with broken corners, impurities such as flint and clay, and those that are undersized, are rejected. The satisfactory bricks, in groups of five, are placed on an inclined roller conveyor. They are conveyed by gravity to a point where they are placed in heavy paper cartons, sealed, warehoused, or loaded directly to freight cars or trucks.

PRESSED BRICK

Two types of pressed brick are manufactured in the brick plant, a simple aggregate and an aggregate containing a binder. The preparation for pressing is identical except for the addition of the binder.

The crude diatomite is hauled to the crushing plant by truck from the quarries. It is fed by Fresno scraper or by hand into a high-speed bar mill with coarse grid openings. The coarsely milled crude is conveyed by rubber belts to a cylindrical revolving screen. The oversize is returned to the crusher by belt conveyor, the undersize fed to the pugger, where about 150 per cent of water is added. The mixed aggregate and water passes into either of two Fernholtz presses with mold boxes for four bricks each. Poorly pressed bricks are returned by bucket elevator to the pugger.

The pressed bricks are placed by hand on tunnel-drier cars in a definite pattern to receive the most efficient drying. The cars are passed through the tunnel driers at regular intervals, and the moisture is completely removed.

The thoroughly dried bricks are treated in two ways. The simple aggregate brick is usually fired in a tunnel kiln; the bonded aggregate brick is fired in down-draft beehive kilns. The calcined bricks, now slightly

pink, are unloaded by hand to a gravity roller conveyor where they are inspected prior to packing. The satisfactory bricks pass on down the conveyor to a point where they are packed, 25 in a corrugated paper carton, and then warehoused. Each brick is examined for physical defects, by the inspector, and sample bricks from each day's production are tested. They are sold as Sil-O-Cel C-22 Brick.

The bonded dried bricks are placed by hand in the beehive kilns. The temperatures are gradually increased to the proper conditions, which include a soaking and cooling period.

These bricks are white, strong and also of standard size, 9 by 4½ by 2½ in., having shrunk exactly 10 per cent. They are known as Sil-O-Cel Super Brick, and are used for insulation where the temperatures are not higher than 2500° F.

All of the kilns and tunnel driers are fired with natural gas. The waste heat from the cooling zone of the tunnel kiln aids in the drying of the pressed and also natural brick.

AGGREGATES

Natural

Crude from quarries is ground in a bar mill of local design. The bars are attached rigidly to a shaft, which turns at about 1800 r.p.m. The material is fed into the mill, and contained therein until of small enough particle size to pass through the grid bars at the bottom. The product is mechanically screened, the fines going to a packer bin and the oversize returning to the mill. The product is sacked by gravity into burlap bags, a full sack weighing 100 lb. The product known as Sil-O-Cel Coarse Grade or Building Insulation is especially suitable for insulations where the temperatures are not higher than 1600° F.

Calcined

Crude diatomite, hand-quarried, piled in rows and air-dried, is fed into a natural gas-fired vertical kiln 60 ft. deep. It is calcined and the finished product is drawn off at the bottom by gravity. A hand-trammed steel mine car carries the calcined diatomite to a bin, where it is stored until ready for crushing at the mill in Lompoc. The same equipment is used as for the milling of the natural aggregates.

The product, known as Sil-O-Cel C-3, is suitable for insulation at temperatures as high as 2000° F.

The vertical kiln is unique in that it is a shaft 12 ft. in diameter sunk in solid stratified diatomite. It is lined with firebrick and the whole hill-side acts as an insulation medium.

PRODUCTS AND PRINCIPAL USES

Filter Aids

Three powders for filtration work are dried, milled and processed; namely, Filter Cel, Standard Super Cel and Hyflo Super Cel.

Filter Cel is natural, dried, milled and processed Celite. It is slightly buff colored, light weight, soft and generally chemically inert (except with hydrofluoric acid and strong alkalis, which dissolve all diatomites). Its chief use is where a high degree of clarification of a liquid is the prime requisite.

Standard Super Cel is dried, milled, processed and calcined in a rotary kiln at elevated temperatures. The resulting powder is light weight, cream to slightly pink in color, chemically inert, and from which the chemically combined water has been removed. Its chief use is as a filter aid, on liquids with larger suspended particles that do not necessitate the fine screening produced with Filter Cel. The rate of flow of filtrate is nearly twice that when Filter Cel is used.

Hyflo Super Cel is treated exactly like Standard Super Cel, except for the addition of small amounts of an alkaline flux. The powder is light weight, dead white, chemically inert. It is used as a filter aid where speed of filtration is the desired object with satisfactory clarification. It is also used on liquids that cannot be filtered with the other products. The flow rate of filtrate is three times that of Standard Super Cel.

By special refining are obtained variations of these powders for special filtration uses.

Admixtures and Fillers

The same general processes are used to manufacture various types of powders for light-weight mineral fillers, for large variety of use, such as admixtures in asphalt, and as admixtures for concrete to make that material more workable and smooth. The white powders are used in paints of various kinds, the finer processed grades for polish.

Insulation

Natural powders and calcined powders are used as insulation products, where heat losses from high temperatures are to be decreased. The powders are milled and processed in much the same way as those mentioned. Molded and pressed bricks as well as natural bricks can be made into any special shape that may be required for the insulation of special equipment.

Coarse aggregates for insulation have been mentioned previously. The natural Sil-O-Cel brick are used for insulation where the temperatures

do not exceed 1600° F. The Sil-O-Cel C-22 brick are used for insulation at temperatures not exceeding 2000° F. The Sil-O-Cel Super Brick are used for insulation at temperatures not higher than 2500° F.

Briefly summarizing, the principal uses of Celite diatomite products are in insulation at comparatively high temperatures and also low temperatures, building insulation, in filtration of all types of liquids from the most viscous to the least viscous; as admixtures and mineral fillers where a chemically inert light-weight mineral is required, and as a mild abrasive in polishes and a filler in paints.

RESEARCH AND DEVELOPMENT

In the early operations the diatomite was simply mined, air-dried and shipped in lump form. New uses being established, it became necessary to pulverize the diatomite and sell it as a powder.

The powdered material gradually assumed more uses and the Celite Company decided that a research department would quickly evolve new processing methods and new products, and at the same time provide technical control of the operations to insure uniformity. A small technical force, primarily interested in technical control, was established. The effects were immediately shown by a closer control of the milling and consequent improvement in products.

Methods were evolved at the same time for milling and drying, and the wasteful method of air-drying of crude was abandoned. Studies of milling and air separation produced extremely efficient blowers, cyclones and dust collectors. The two major calcined powders were planned and produced successfully to perform many filtrations more economically and to permit filtration of liquids unfilterable formerly. All new processes for milling were examined in great detail before being built into the various production systems.

The sawing of natural brick in the quarry was a result of considerable thought, as was the establishing of higher temperature insulating brick.

In 1928 the technical force at Lompoc numbered 30. Departments had been created covering the inspection and control of finished products, geological control of the crude diatomite, research in admixtures for concrete, asphalt and rubber, research in matters pertaining to filtration problems of all kinds, research in insulation as bricks, molded blocks and powders, research in polishes, and research in milling and air separation equipment.

In 1929, Johns-Manville centralized its entire research staff at Manville, N. J., where elaborate laboratories were established for research work on all products manufactured by the Corporation. The Celite Research Department was transferred from Lompoc, except the section devoted to inspection and control and plant development.

The development from crude milling to the highly improved milling and air separation now used, and the great numbers of varied new uses are largely due to the efforts of the technical department.

POWER AND FUEL

Electric power is transported by high-voltage lines from a substation at Lompoc and is furnished by the Midland Counties Public Service Corporation. Natural gas for drying and calcining is furnished by the Santa Maria Gas Company.

MAINTENANCE

The deposit is so far from large cities that it is necessary to have trained machinists, blacksmiths, carpenters and electricians, as well as a fully equipped machine shop at the plant.

Nearly all of the milling equipment is a result of local development, and so all cyclones, separators, blowers and crushers are manufactured at the plant. Maintenance on all equipment, buildings, roads, etc., is taken care of by this department.

CAMP AND HOSPITAL

There are 154 houses in the hills close to the milling plant. They are primarily for the daily payroll men and their families, and are rented at a nominal fee per month. All are equipped with electric lights, running water, and in special cases with natural gas. Many of the employees avail themselves of the opportunity for low rent and proximity to the works. Others reside in Lompoc and commute by automobile.

A general store is maintained by private interests in the center of the main camp. A school for the first six grades is a short distance away, in San Miguelito Canyon.

A full equipped hospital in the main camp includes a sick bay, operating and complete X-ray equipment. Usually only first aid is rendered here, the more serious cases being removed at once to Santa Barbara or even Los Angeles.

SAFETY WORK

The company and its employees are safety conscious to a remarkable degree. A weekly inspection of all operating equipment is made by committees composed of three workmen and the leader in this work. The reports of these committees are carefully examined and faulty equipment, poor guards, or improper working conditions are changed or corrected. The personnel of these committees is changed weekly. Owing chiefly to these weekly examinations by the workmen themselves, and to the fact that the company immediately takes steps to remedy all faulty

equipment and to place effective guards where needed, serious accidents seldom occur and minor accidents are few.

SUMMARY

The deposit of diatomite covering 4000 acres, and more than 1000 ft. thick, is composed of Marine diatoms assigned to the Monterey formation of the Upper Miocene.

The main deposit and workings are confined to a broad, gently dipping syncline, both limbs of which are being quarried. At present all mining operations are confined to the surface where gasoline and Diesel shovels quarry the diatomite directly into trucks for transportation to vertical storage shafts. Diatomite destined for the mill and rotary kilns is hauled underground by an electrified line.

Natural bricks of standard size are sawn directly in the quarries by machines designed by the company's engineers.

Pressed bricks are manufactured for insulation at high temperatures.

Owing to the intensive work of a large staff of research men, many new products have been developed as well as hundreds of new uses. The machinery and equipment for milling diatomite have been the result of local development.

ACKNOWLEDGMENTS

Nearly all of the information contained in this report is a compilation of data from the author's intimate contact with the geological, mining, milling and research phases of the diatomite operations at Lompoc while employed by the Celite Company and the Johns-Manville Products Corporation.

For a considerable portion of the historical sketch the author is indebted to Mr. A. G. Balaam, of Lompoc. Further information regarding the early period was derived from *Bulletin* 38 of the California State Mining Bureau, 1906, on The Structural and Industrial Materials of California.

Acknowledgments are also due to Mr. A. Cromwell, Mr. W. B. Kelley, Mr. A. S. Elsenbast and Dr. A. B. Cummins, of the Johns-Manville Corporation, for their interest and cooperation.

DISCUSSION

(E. F. Burchard presiding)

W. W. BRADLEY,* San Francisco, Calif.—Why is there abrasion on the cyclones and pipes? I think you also told me at the plant one day that there is a rather high abrasion on the teeth of the saws with which you cut the brick.

H. MULRYAN.—In handling 500 or 600 tons at times—an average, say, of 300 or 400 tons a day of siliceous material—there is bound to be some abrasion. It is a very

* State Mineralogist of California.

mild abrasion, however. Polishes made from diatomite all pass through a 325-mesh sieve. It is a good deal softer than any of the usual abrasives. The chief use of the material produced from the natural powder is in automobile polishes. The baghouse material from calcining operations is a little more abrasive and is used in metal polishes.

MEMBER.—What gives the material its filtering qualities? I understand it is much greater than any other material available. Is it a mechanical or a chemical action or just what?

H. MULRYAN.—There is no chemical action. In a filtration system a filter press is used under pressure, with a backing of some kind of cloth. In order to keep the porous spaces from becoming clogged, a filter aid is used that forms a mat on the outside of the cloth. The spaces are kept open for a considerable period of time. The filtration is purely mechanical.

J. A. CARPENTER,* Reno, Nev.—I am very much interested in that 36-ft. diameter and 50-ft. deep shaft. It was probably driven by going up from the tunnel, was it not? How did you drill and blast in that material?

H. MULRYAN.—That shaft was raised 80 ft. vertically from the tunnel level, including a manway in the north wall. Because of poor ventilation and other difficulties, it was decided to sink the remainder of the shaft from the surface. The surface location was made by instrumental survey. The shaft is probably unique because of its method of sinking.

We had a haulage tunnel, 80 ft. of shaft, and bin gates for the removal of the excavated material. A Keystone well drill rig was used to sink a 12-in. hole to the top of the raise. Then a reamer was designed with increasing diameters from 10 to 36 in. at the widest point. The reamer was made of 12-lb. rails chisel-pointed on the ends, the entire basket of rails being welded to a regulation drill stem. The 12-in. hole was reamed to 36 in. for the entire depth, using the same Keystone drill rig. The excavated diatomite dropped through the hole into the raise, where it was drawn off into electrically powered tramcars and hauled to the waste dump.

An air hammer with Carr bit was used to drill holes 5 ft. deep around the 36-in. hole in circular form 5 ft. apart and 5 ft. away from the small hole. Three circles of holes were drilled, loaded with 20 per cent dynamite and detonated with No. 8 electric delay caps. The diatomite was shot into the 36-in. hole in the center and drawn off in the tunnel.

A timbered manway was sunk in the north wall to its connection with the previously raised manway. Later the diameter of the shaft was increased to 36 ft. to provide additional storage of diatomite. Holes were drilled as before around the circumference of the 18-ft. shaft, loaded, shot, and the waste removed as before. The manway was removed in this operation.

The total depth of the shaft is now 350 ft.; the diameter, 36 ft. There is no timbering, no supports other than the concrete bottom in the tunnel.

A. H. KRIEGER (written discussion).—This paper by Mr. Mulryan contains some minor inaccuracies and a few omissions. Under the title of "History" two men, Hannaman (misspelled Harriman) and Hanmore (misspelled Hanimore) are entitled to wide credit. George B. Hannaman spent 14 years of a vigorous and searching life on these Lompoc properties and it was his vision and imagination that brought the Balaam ranch mentioned in the paper and four other ranches together under one ground ownership. A difficult piece of work! Hanmore is the man who invented the 85 per cent magnesia with 15 per cent asbestos combination, which has had by all odds

* Professor of Mining, Mackay School of Mines.

the widest distribution for steam insulation in this country. He experimented in the early days with diatomite from Lompoc to take the place of magnesia and he made mixtures for insulation that are coming into use 40 years after his death.

In 1912 Messrs. Fitger and Anneke and Krieger, of Duluth, Minn., acquired title to the ground area from the Magne Silica Co. They did not change any name but organized as Kieselguhr Company of America, because what was known of the material in those days was described under the generic name Kieselguhr. As soon as operations began at Lompoc in 1912 a staff of scientific, research engineers was engaged under the leadership of Percy Boeck, and the markets were searched for outlets. Products made for different uses were put out under trade names and "Celite" was coined as the mother word for the material as mined or quarried and from which the market grades were manufactured. At the suggestion of Clowry Chapman, sales consultant and trade mark expert, the difficult and generic term Kieselguhr was dropped for the protected, euphonious name of Celite, which is still used by Johns-Manville.

What the author describes as a strong effort to have diatomite used for the filtration of sugar was so successfully done under the guidance of Walter Lyle Jordan that in a few years, from 1914 to 1921, almost the entire sugar world was filtering its liquors with diatomite from Lompoc. This use for Lompoc diatomite has greatly lessened in recent years because of severe and intelligent competition.

It is certain that at varying intervals during the depositing of these huge beds of diatomite great masses of clay were swept into this basin and all prior deposits of diatoms were buried. These beds of clay vary from a fraction of an inch to over 100 ft. in thickness. The 1000-ft. section described by the author as "soft beds" contains these clay strata and hundreds of feet of other material hard and not marketable. None of the Lompoc diatomite is white. At best it is light gray, and when wet it ranges from light to deep brown.

The mining methods employed now and explained on pages 485 and 486 are the last word in large organization planning and execution. Of interest to open-cut mining engineers were the electrically driven cutters handled by one man, which channeled kerfs from 3 to 4 ft. deep at right angles to the strata and made blocks 3 to 4 ft. square. These blocks were lifted out by stiff-leg derricks and hauled away. And there was a singular device of a long, endless chain, with cutters fastened into the links, which was drawn across the toe of a hill near a quarry and made cuts over 20 ft. deep. This method sliced the hill in parallel cuts 4 ft. apart and permitted very inexpensive quarrying of large blocks of "chalk rock," the local misnomer for diatomite. Frequently these blocks were hauled to a sawmill much like a lumber mill, where they were put through a continuous operation of sawing until they came out 2½ by 4½ by 9 in.—firebrick size.

The phenomenal calcining kiln built into the hillside 20 years ago is still in service without improvement. It was an upraise, in No. 9 diatomite quarry, enlarged and shaped like a vertical shaft kiln and holing out in the quarry floor. A gallery around the bustle contains the firing apparatus and another gallery around the base serves as a drawing-off floor with a tunnel connection and a tramway to the crushing plant at the mouth of the tunnel on the side of the hill below. The semi-air-dried material is fed into the top of the kiln from the open quarry floor.

To engineers everywhere it is of interest that the Lompoc diatom beds supplied material for the first high heat insulation. It was marketed under the trade name Sil-O-Cel. This product entered a new field never before attempted and its development opened up the entire field of high-temperature insulation and control. It was tested for and used in boiler walls, hot-blast stoves, hot-blast mains, by-product coke ovens, glass tanks, heat-treating furnaces, enameling furnaces, brick and tunnel kilns, oil retorts and gas plants, firebox pans on navy vessels, soaking pits, electric furnaces, gun furnaces and hundreds of other applications.

The powders, aside from sugar filtration, went into battery boxes, matches, hard rubber, paints, wall blocks, plaster, concrete, brick mortar, cold-storage walls and ceilings, absorbents, etc., etc. In the filtration field every product came under the effect of diatomite where clarity was important.

The geology of diatomite at Lompoc is still unanswered. How it was laid down, whether the bottom of the basin in which it occurred was flat or wavy or whether the uplifting changed the contours to the angles we find there today are interesting and should be studied. Mr. Mulryan should be permitted to finish a job well begun.

H. MULRYAN.—Mr. Krieger's reference to the original owners and the correct spelling of their names are points well taken, because the names appear as Harriman and Hanimore in "Structural and Industrial Minerals of California," *Bull.* 38, California State Mining Bureau, page 296 (1906).

His discussion relative to clays in the "soft beds" is based on incorrect information. There are no true clay beds in the deposit at any place. The most impure beds seldom contain as much as 10 per cent alumina and they are composed of diatoms, sponge spicules, silicoflagellae, volcanic ash, and extremely fine-grained silt. The impurities are removable by present-day methods and the beds are now considered usable for some purposes. These impure beds are referred to on page 475, line 36, as containing argillaceous material. There are no great masses of clay, and the indications, geologically, point to an area of sea water in which there was very little inrush of sediments; in fact, to an area of water that was very quiet and little affected by erosional forces.

I do not state that the diatomite is white. On page 477, line 29, my reference is to the thinly laminated streaks that are actually white. In line 35, reference is made to the true color of the thick beds of diatomite as "slightly gray or even light brown."

No data were given on the channeling machines because of small space allotted to history. They are practically unused at the present time.

The calcining kiln is referred to on page 492, lines 30 to 40, and I believe in sufficient detail because of its relative importance.

The references to insulation are duplicated in the short paragraph on page 493.

The geology of the beds at Lompoc is rather definitely known, as the strata are exposed in detail. It is generally admitted that the siliceous beds were deposited on the older formations in a horizontal manner, and that their present attitude is due entirely to dynamic movement resulting in the present physiographic features.

W. L. JORDAN, New Brighton, Staten Island, N. Y. (written discussion).—In 1912 samples of the Lompoc deposit were brought to the attention of Arthur Krieger, of Duluth, Minn., who had been engaged in the production and distribution of asbestos and magnesia insulating products for a number of years previously. Mr. Krieger appreciated that this Lompoc diatomite, by proper exploitation, could produce products that would perform a distinct service to industry.

Standardized products produced for specific uses and marketed under appropriate trade names, market control by direct sales to the consumer and serviced by an engineering sales organization were the high spots in the campaign that in eight years increased the production and consumption of diatomite materials in this country from 1000 tons to 60,000 tons per year. Research of the highest order contributed to this achievement.

A. B. CUMMINS,* Manville, N. J.—In getting together his paper, Mr. Mulryan was not intentionally overlooking or ignoring historical aspects of the development of the deposit. In the short compass of the paper he had to try and set up the present-

* Johns-Manville Research Laboratories.

day industry, describe modern developments, and give us an over-all view of what the industry is today, and we should not feel that he has in any way endeavored to minimize the work of the past or historical aspects of the subject.

E. SAMPSON,* Princeton, N. J.—What are some of the other materials that have become competitors with diatomite? What other raw materials have cut into this market?

E. F. BURCHARD.—I do not know anything in the solid block form. I suppose tripoli may be a competitor in a small way.

A. B. CUMMINS.—Perhaps we may misunderstand the drop in tonnage. As I recall the figures, the American diatomite production in 1929 was 40 per cent over any preceding year.

The chairman has expressed it very aptly. There is not any outstanding competition to diatomite as a raw material. In all of its different uses, it does find competition, or it meets competition either in the processes or in some specific use, but in its over-all aspects I think we could best answer your question by saying that the use of the material is going forward rather than the reverse. Conditions vary from year to year, and the percentage of the material used in one industry or another also varies.

A. KRIEGER.—A great deal of the tonnage in diatomite does not come from Lompoc. I think Lompoc was a monopoly in 1928.

A. S. ELSENBAST,† New York, N. Y.—I would like to clarify the situation as far as the tonnage is concerned. We have competition from other processes and materials that are not directly related to the product itself. In addition, there is a very large loss due to regeneration of the material in certain industries. Also, there is a decreased use of the material as a result of more efficient use of the material and a very greatly reduced consumption due to a more efficient product. In other words, we are now more efficient. We have filter grades, 10 per cent of which in certain cases will do the work of 100 per cent of that material originally, and as a result, while the industry as a whole has gone ahead and is on a sounder basis, we have had large losses in certain directions, which have had to be made up by other uses.

* Geologist, Princeton University.

† Johns-Manville Corporation.

Tripoli Deposits of the Western Tennessee Valley

BY E. L. SPAIN, JR.,* JUNIOR MEMBER A.I.M.E.

(New York Meeting, February, 1936)

THE deposits described in this paper occur over much of Wayne County and in the southeast portion of Hardin County, Tennessee, and in the northeast and northwest portions of Mississippi and Alabama respectively. The presence of tripoli in this region has been known for some time and descriptions of various occurrences in Tennessee have been made by Jewell¹ and Miser², while Hilgard³, Lowe⁴ and Logan⁵ have treated the deposits that occur in the adjacent portions of Alabama and Mississippi, but there has not been a general survey of the entire field summarizing completely the geology of the deposits with respect to their origin and stratigraphic relationships. During 1934, while engaged in surveys of mineral resources through this region for the Tennessee Valley Authority, the writer collected data with this in mind, and has assembled them in this report. He is indebted to H. S. Rankin and N. A. Rose, of the Tennessee Valley Authority Geology Division, for field assistance.

LOCATION OF TRIPOLI DEPOSITS

Fig. 1 shows the locations of deposits of tripoli in a long belt that extends from northeast Wayne County southwestward into northwest Alabama and thence passes into Mississippi, where the formations carrying it are finally covered by Cretaceous sand and gravel of the Mississippian Embayment. Occasional deposits occur also in the southeastern corner of Hardin County, Tennessee. While tripoli is found at nearly all points along this axis where the related formations are exposed, the largest deposits are grouped about three different centers.

Waynesboro-Collinwood District.—In the area between Waynesboro and Collinwood, in Wayne County, Tennessee, tripoli of various grades outcrops in the valley of Green River and in tributary valleys on either side. This district is roughly 12 miles long in a northeast-southwest direction and has a maximum width of 5 miles at its northern end. Exposures are found in road cuts, highway chert pits, in the beds of some of the streams. and on hillsides. The greatest number of high-grade

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¹ References are at the end of the paper.

deposits are found in this area, which in addition is most accessible, being served by a branch line of the Louisville and Nashville Railroad to Col-



FIG. 1.—MAP SHOWING LOCATIONS OF TRIPOLI DEPOSITS OF THE WESTERN TENNESSEE VALLEY, WITH STRUCTURE CONTOURS DRAWN ON CHATTANOOGA SHALE.

linwood, and by surfaced state highways, one of which traverses the entire length of the outcrop between Waynesboro and Collinwood.

Stout District.—In the vicinity of Stout, in the extreme southwest corner of Wayne County, tripoli occurs abundantly in Wayne and Hardin counties, Tennessee, and in Lauderdale County, Alabama, an area comprising approximately 30 sq. miles. While not as accessible as the Waynesboro-Collinwood deposits, the Florence-Savannah highway (Ala. No. 2, Tenn. No. 69) now under construction passes through their center and connects with the closest railroad at Florence, Ala., 25 miles away.

Bear Creek District.—Tripoli occurs from Riverton, Ala., southward along the east side of Bear Creek almost to Margerum, and at several localities on the west side of Bear Creek in the adjacent portion of Tishomingo County, Mississippi. All of these occurrences may be reached by surfaced county roads from Margerum, Ala., and Iuka, Mississippi.

TOPOGRAPHIC RELATIONS

The tripoli occurs in the southwestern part of the Highland Rim Province of Tennessee, a more or less flat plateau underlain by siliceous limestones of Mississippian age. In this area, because of the proximity of the Tennessee River, the plateau has been highly dissected by stream action into narrow, twisting flat-topped divides between the steep-walled valleys. The larger streams have cut broad, open valleys and a few of them, like Bear Creek, which has cut almost to base level, meander widely. Smaller tributary streams, however, occupy youthful, V-shaped valleys.

Maximum relief in the northern portion of the tripoli belt averages 300 ft., the general elevation of the plateau lying at elevation 1000 or more. To the south in Alabama and Mississippi this relief increases to 400 feet., with the top of the plateau lying slightly lower, since the region slopes gently to the southwest.

It is upon the sides and in the bottoms of the stream valleys of this dissected region that the tripoli occurs. It lies well below the general plateau crest, and where it is exposed along the larger streams that have cut deep valleys it is usually high above drainage level. Elevation of the base of the tripoli horizon in the Waynesboro-Collinwood district is at approximately 800 ft. In the Stout district elevations average 625 ft., while in the southernmost Bear Creek district the average elevation is 425 ft. The total drop in the 40-mile extent of the tripoli, therefore, is approximately 400 ft., or well less than 1° . This general slope is flatter, but otherwise corresponds to the regional dip of the rocks in which the tripoli occurs.

Tripoli outcrops along valley sides have a marked effect upon the topography, in many places forming nearly vertical banks. Although in no sense consolidated, the tripoli is heavily compacted and tends to maintain the same steep face that was cut by the stream that exposed it.

The material that overlies it, however, is loose, coarse residual chert, which seeks a normal angle of repose. Even though exposures may be hidden by vegetation and humus, this topographic break indicates the outcrop.

STRATIGRAPHIC RELATIONS

A generalized outline of the stratigraphy of the tripoli districts is given in Table 1. The formations directly involved will be described briefly.

TABLE 1.—*Generalized Section of Area in Which Tripoli Occurs*

		Ft.
Pleistocene and Tertiary (?)....	Terrace gravel and sand, stream alluvium.	0- 20
Cretaceous system.....	{ Eutaw sand.....	25- 50
	{ Tuscaloosa gravel and sand.....	50-150
	{ Warsaw and St. Louis limestones.....	50-200
Mississippian system.....	{ Fort Payne limestone.....	50-150
	{ Ridgetop shale.....	0- 80
Mississippian or Devonian.....	Chattanooga black shale.....	0- 30
Devonian system.....	Olive Hill limestone.....	0- 80

Fort Payne and Warsaw Limestones

The Fort Payne does not crop out in a fresh state, but it has been reduced by weathering to a mass of blocky chert fragments and pulverulent silica. In parts of Alabama and other portions of Wayne County it is a limestone containing layers of chert and argillaceous material as well as lenses of pure limestone. The leaching out of the calcareous constituents of this formation is responsible for its present highly weathered condition and the formation of tripoli at its top.

Since tripoli occurs both at the top of the Fort Payne and the base of the Warsaw, the character of the chert is in most instances the only guide to the identification of the parent formation. It is important, therefore, to note the differences between the residual chert of the Fort Payne and that from the overlying Warsaw limestone, as certain differences appear to hold good over the entire region.

Chert from the Fort Payne is solid and blocky, tending toward a cubical form due to its cleaving at right angles. This cleavable character is so strong that under weathering the fragments break down into smaller and smaller pieces. In the large fragments crinoid stems may be found at certain horizons, although as a whole the Fort Payne is not highly fossiliferous.

Chert from the Warsaw, on the other hand, is highly porous and does not cleave like that of the Fort Payne. This massive, comparatively soft chert is almost always filled with fossil remains and sometimes is entirely composed of matted Bryozoa, the presence of which reveals its identity quickly.

Miser states² that the St. Louis limestone, also of Mississippian age, directly overlies the Fort Payne in Wayne County, the intervening War-

saw being absent. From a study of fossil remains and a careful comparison of cherts, however, the writer finds this succeeding formation to be Warsaw. As with the Fort Payne, no unweathered Warsaw was observed in the region. Elsewhere, where the formation is fresh, it is usually a much purer limestone than the Fort Payne, although it contains some free chert and in many localities it is sandy or siliceous in its basal portion. Nearly all the tripoli deposits examined that showed a high enough quality to be of commercial grade apparently occur in the Warsaw.

Cretaceous and Overlying Rocks

Tuscaloosa and Eutaw Formations.—Over nearly the entire extent of the tripoli belt the tops of the hills are made up of unconsolidated Tuscaloosa gravel, which toward the southern end reaches a thickness of more than 100 ft. This formation consists chiefly of well rounded, equidimensional chert gravels, but contains also some sand and clay.

Up as far as and including the Stout district, the tops of the hills and divides are capped by the Eutaw sand, a cross-bedded red sand formation containing no gravel, but occasional stringers of clay.

Scattered as a thin mantle over the tops of the hills and ridges are flattened quartz and quartzite pebbles of stream origin. Those at the higher levels on the Highland Rim surface are believed to be Pliocene in age.

CHARACTER OF THE TRIPOLI

Physical and Chemical Properties

Tripoli is essentially silica in a very finely divided state, occurring naturally and not produced by the grinding of quartz, flint or any of the more massive forms of silica. It is derived through the weathering and decomposition of a siliceous limestone or a calcareous chert. Two types of tripoli are produced in this country:

The Missouri-Oklahoma type occurs at various points in Missouri and Oklahoma, centering about Seneca, Mo. In color it varies from cream to deep red. It has a grain size of 0.01 mm. or less, and is a more or less coherent material, which is quarried in blocks and subsequently ground.

The Illinois-Tennessee type is dense and coherent in the Illinois phase, but at Butler, East Tennessee, the locality of the Tennessee type, it is entirely noncoherent. The grain size of this type is much smaller than the Missouri-Oklahoma material, being 0.002 mm. or less. The color is usually white or only light cream. It is widespread in occurrence, and it is to this division that the West Tennessee tripoli belongs.

Tripoli from the area under discussion is usually pure white in color except for occasional brown streaks caused by iron oxide. It is very soft, particularly when wet, and may easily be reduced to powder by the fingers. It is difficult to remove from the hand after crushing, because of its

fineness of grain, and owing to this characteristic is locally termed chalk. The dry powder, even though crushed only by the fingers, will easily pass a 100-mesh sieve, and a small amount will pass 200 mesh. When dry it is very light, and the specific gravity varies between 2 and 3.

There are differences between the tripoli derived from the Warsaw and that from the Fort Payne. Under the microscope the grains of that from the Fort Payne are seen to be larger and more fragments of unweathered chert are mixed with the true tripoli. The grains of the Warsaw material are of fine, even size with a pronounced irregularity of shape. Under low power they appear to be amorphous, but when magnified to 500 diameters, approximately one-third of them show distinct double refraction, proving their crystalline character. Aside from the fact that the fineness and evenness of grain makes the Warsaw tripoli of higher quality than that from the Fort Payne, the Warsaw deposits as a whole are more free from the undecomposed chert that marks the Fort Payne deposits. This high percentage of undecomposed chert often excludes deposits from any commercial consideration because of the waste that would be involved in mining.

Some of the tripoli is stained by iron in places and the microscope shows that the individual grains are coated and permeated by this thin, brown stain. In addition, there appears to be a flocculent, brown iron compound dispersed through the interstices of the mass. As a general rule, this staining is only surficial. After digging back a few feet at the most in an exposure, the tripoli becomes white, showing that oxidation proceeds slowly back from the surface. In this connection, when drying the material before crushing, care should be taken not to raise the temperature too high, since this oxidation will be reproduced in the process and the product will be colored. In the laboratory many samples of pure white tripoli quickly turned brown at red heat.

TABLE 2.—*Analyses of Tripoli from Hardin County, Tennessee^a*

Locality	Composition, Per Cent					
	SiO ₂	Al ₂ O ₃	Fe ₂ O ₃	CaO	MgO	Ign.
Upper Horse Creek.....	96.92	1.68	0.40	0.10	0.03	0.78
Scott Hollow.....	95.89	2.55	0.53	0.11	0.02	0.98
Head of White Creek.....	92.37	4.64	0.26	0.08	0.02	2.46

^a From W. B. Jewell¹.

Chemically tripoli is nearly pure silica. Various analyses from this area show a silica content from 92 to 99 per cent, with small amounts of Al₂O₃, Fe₂O₃, CaO and MgO in the order of their abundance. The tripoli that has been produced from these districts contains 98 per cent

or more SiO_2 . The usual ratios of the constituent compounds are illustrated in the analyses of Table 2.

Form of Deposits

The tripoli lies in essentially horizontal beds, which show a wavy, uneven, lamellar structure. Solution and resulting slump structure often throw these beds into sharp folds, the intensity of which shows that a large amount of calcareous material has been removed. Outcrops are frequently intermittent within the same deposit. The thickness varies and in places the beds may be almost discontinuous (Fig. 6). Very steep banks are formed at the outcrop, usually free of vegetation because of occasional slides, which break a new surface on the exposure. In the Stout district such a slide revealed a 60-ft. vertical face of tripoli. In the many occurrences examined during field work, the thickness of tripoli beds varied from only a few feet up to 60, the average among the larger deposits being about 25 feet.

Exposures of tripoli in the Warsaw are usually smoothfaced and more nearly vertical. The Fort Payne exposures, on the other hand, are rough and cherty as a rule, and while chert layers occur in both types, they are much more abundant in the Fort Payne. It is difficult to determine the areal extent of tripoli deposits. Because they are almost always exposed on steep hillsides or valley walls, it is impractical to drill the thickness very far back of the face, owing to the heavy chert and gravel overburden. At Eastport, in northeastern Mississippi, in a drift 300 ft. back from the face the tripoli is still of unchanged good quality. The mine in Hog Hollow, near Collinwood, Tenn., has a drift 100 ft. long driven into a 300-ft. exposure.

Origin

The tripoli is the residual product of solution of calcareous matter from the Fort Payne and Warsaw limestones. The tripoli belt corresponds closely with an axis of the Nashville dome (Fig. 1), the structure contours being drawn upon the surface of the Chattanooga black shale⁶. The overlying formations are particularly well exposed to general weathering processes having been more or less broken along this anticlinal axis, and it is here, along the main axis and on divergent structural spurs, that the most complete weathering has taken place in the region. It is probably this structural condition that has resulted in the formation of such large amounts of tripoli here while there is a comparative lack of it on either side. While this is perhaps a generalization, observations at nearly all points bear it out.

The chief difference of opinion with regard to the formation of tripoli is whether the chert left after the leaching of the limestone breaks down further to form tripoli or the tripoli is the direct product of siliceous

limestone decay. There are indications that both processes have taken place in this field.

Formation of tripoli from chert appears to have taken place in the Fort Payne. Pieces of chert are found which are rounded by attrition, and former outlines of rough chert blocks are sometimes faintly apparent in the present tripoli. This material, however, is of a very inferior quality.

Siebenthal⁷, postulated the direct formation of tripoli from a finely siliceous limestone, which leaves a porous skeleton of silica behind after solution. There is strong evidence in support of this process and it is apparent that nearly all of the tripoli in this region owes its origin to it.

The tripoli is very homogeneous. Smoothness of bedding and well defined lamellar structure is the rule, and there is no irregularity such as

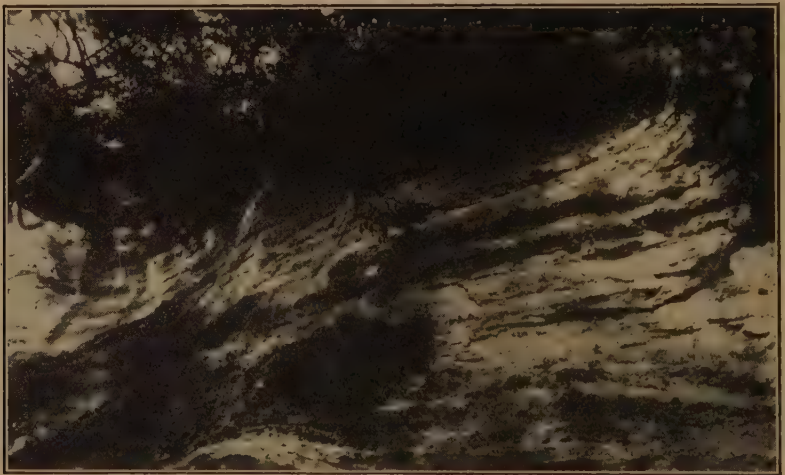


FIG. 2.—OUTCROP OF TRIPOLI NEAR STOUT, TENNESSEE.

Shows fine, even bedding and absence of any distortion, which would be produced if the tripoli were derived from a mass of chert blocks.

would be produced had it been formed from a mass of jumbled chert blocks (Fig. 2). The chert layers show fragments that are sharp and unweathered and only in a few scattered instances has it been possible to say that there is any gradation between the two. The deposits near Stout contain layers of chert enclosed above and below by the finest of tripoli with no gradation between (Fig. 3). In addition, no fossil casts have been observed in the Warsaw tripoli such as would have been represented had the highly fossiliferous chert decayed in place.

The Warsaw limestone elsewhere in Tennessee is siliceous in its basal portion. On the Eastern Highland Rim of Tennessee thick beds of fine sand often result from weathering. This basal siliceous member is also present to the north on the western rim and undoubtedly extends into the tripoli field. In 1934 the Tennessee Valley Authority drilled a quarry

site near Sheffield, Ala., which lay within the Warsaw formation and complete analyses were made of the cores. While the upper part of the formation was remarkably pure limestone, averaging 99 per cent or more CaCO_3 , the silica content, which averaged less than 1 per cent in the upper portion, increased to an average of 30 per cent as the basal portion was approached. The limestone, which had been crystalline and oolitic at the top, became a dull gray, fine-grained rock. Samples from this lower horizon were subjected to leaching by dilute hydrochloric acid and a porous mass of silica was left, which could be crushed to a fine powder by



FIG. 3.—LAYER OF UNDECOMPOSED CHERT IN A FACE OF TRIPOLI.
There is no gradation between the chert and the fine tripoli above and below.

the fingers. This siliceous member of the Warsaw persists throughout the entire region in which the tripoli occurs, and it is believed that the main body of tripoli owes its origin to the leaching of this siliceous limestone.

THE TRIPOLI DEPOSITS

While nearly 100 deposits and occurrences of tripoli were observed through the region, detailed work was done chiefly on the larger ones. The deposits represented on the location map (Fig. 1) are not distinct and separate deposits, but rather a concentration of them. Typical deposits from each of the three districts are described below.

Waynesboro-Collinwood District—Hog Hollow

This deposit is 4 miles south of Waynesboro, on the Collinwood road. It lies in the head of Hog Creek, a small tributary to Hardin Creek, which is a major stream of the region. Tripoli outcrops for more than 300 ft.

along the south valley wall, and auger holes have found a consistent average thickness of 30 ft. Fossil and chert evidence shows this deposit to be entirely within the Warsaw.

Tripoli in this deposit represents the highest grade of any in the region. It is homogeneous, fine-grained, and perfectly white in color. From the latter part of 1933 up to the present time a mine has been

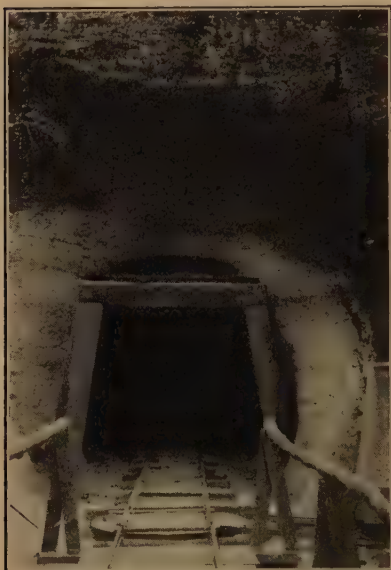


FIG. 4.—ENTRANCE TO TRIPOLI MINE OF CENTRAL SILICA CORPORATION NEAR COLLINWOOD, TENNESSEE.

operated here intermittently by the Central Silica Corporation of Birmingham, Ala. A drift has been driven 100 ft. into the hill and a side drift 75 ft. long has been dug from the main one. Except for 12 ft. at the entrance, these tunnels are not timbered but are arched and self-supporting (Fig. 4).

The crude material is soft and is easily mined by placing light charges of dynamite in auger holes drilled into the tunnel heading. The tripoli shot down is loaded by hand into tramcars, which are pushed out and dumped into a loading bin at the entrance to the mine. Since there is little coarse chert present it is easily avoided in such mining and, as a consequence, little waste material is sent to the mill. From the mine the tripoli is trucked to the mill at Collinwood, 6 miles away, where it is

placed in a storage shed. The mill, a small, but modern and efficient plant, is operated by a 200-hp. Diesel engine and has a capacity of about 40 tons per day. It is similar in design and process to the mill operated at the silica mine near Butler, East Tennessee, some years ago.

Since the crude tripoli is noncoherent, it needs no preliminary crushing, but is conveyed directly into a 4 by 20-ft. wood-fired rotary kiln. After drying it is fed into a 2 by 6-ft. Hardinge pebble mill, flint-lined. Upon leaving the mill the ground material passes through a $\frac{1}{4}$ -in. mesh trommel where the waste, consisting of large, unweathered chert fragments, is eliminated. It is then elevated into an 8-ft. Gayco air separator, from which emerges the finished product (Fig. 5).

With this adjustable air separator it is possible to prepare four different grades of tripoli flour:

1. All passing 60 mesh, 65 per cent passing 200 mesh.
2. All passing 80 mesh, 85 per cent passing 200 mesh.
3. All passing 100 mesh, 95 per cent passing 200 mesh.
4. All passing 200 mesh, 85 per cent passing 325 mesh.

The finished product from this deposit contains more than 98 per cent silica, and, from its appearance in comparison with tripoli from other districts, will compete in quality with any tripoli flour now produced.

Although complete exposures are lacking, there are indications that there is an enormous quantity of tripoli in the Waynesboro-Collinwood district, making it the most important of the three districts under discussion as far as reserves, quality and accessibility are concerned.

Stout District—Second Creek

In the extreme southwest corner of Wayne County, Tennessee, a road cut of the Florence-Savannah highway intersects a large tripoli deposit of very high quality (Figs. 6 and 7). Its greatest measured thickness is 60 ft. and although the total outcrop seems to be discontinuous in places, indications are that exposures are simply covered by chert talus from above. The total length as measured along the highway is approximately $\frac{3}{10}$ of a mile.

A few chert layers present in the face of the deposit indicate its Warsaw origin. The tripoli is pure white, soft, and easily crushed by the fingers (Fig. 8). The grain size is small and uniform. An average of analyses shows silica to be about 96 per cent, iron oxide 1 per cent, alumina and remaining, 4 per cent. On the west side of Second Creek, $\frac{1}{2}$ mile away, tripoli is again found in the road cuts at the same elevation as the deposit described.

There are many other exposures of tripoli in this immediate area, but the one at Second Creek surpasses all of them in quality and size.

Bear Creek District

Tripoli is most abundant at and near the old mine one mile southwest of Eastport, Tishomingo County, Miss. The mine was operated for a number of years until about 1912, when it was abandoned, probably because of adverse economic conditions, as the quality of the tripoli is unchanged in the drift headings. Three large parallel galleries were tunneled into the face for 300 ft. and are connected by numerous cross passages. The crude material was trammed across Bear Creek into

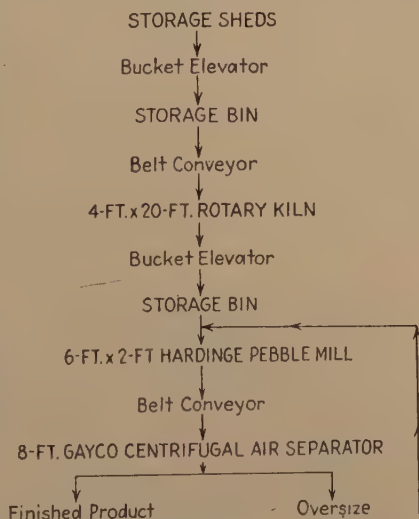


FIG. 5.—FLOW SHEET OF DRY PROCESS USED IN TRIPOLI MILL OF CENTRAL SILICA CORPORATION, COLLINWOOD, TENNESSEE.



Fig. 6.—TRIPOLI DEPOSIT NEAR SECOND CREEK, SAVANNAH-FLORENCE HIGHWAY, WAYNE COUNTY, TENNESSEE.

Alabama to a plant located on the now abandoned Riverton Branch of the Southern Railroad, where it was processed and shipped.

The tripoli in this deposit is considerably more cherty than that in the other two districts, consequently waste must have been somewhat higher in the milling process. The color is white or only slightly stained, and the tripoli itself is of good quality. It is difficult in this instance to say whether the deposit lies in the Fort Payne or the Warsaw and it is possible that both formations have contributed some of the silica.

TABLE 3^a.—*Typical Production and Value of Tripoli*

Year	Production, Short Tons	Value	
		Crude (Estimated)	As Sold (Crude and Finished)
1928	34,043	\$73,689	\$555,578
1929	38,011	74,475	545,658
1930	32,439	71,790	507,505
1931	26,682	56,248	310,131
1932	14,755	31,422	232,700

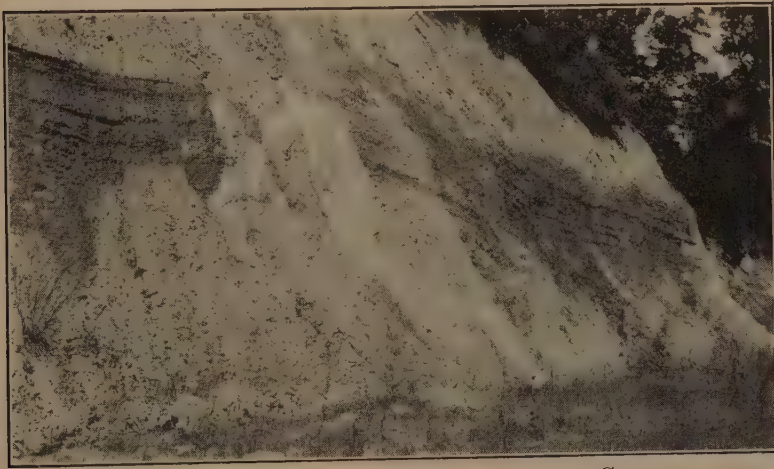
^a From U. S. Bur. Mines Minerals Yearbook for 1932-33.

FIG. 7.—PORTION OF 60-FOOT FACE OF TRIPOLI AT SECOND CREEK IN SOUTHWEST WAYNE COUNTY, TENNESSEE.

TABLE 4^a.—*Uses of Tripoli*

Use	Production, Short Tons	Value as Sold (Crude and Finished)
Abrasives.....	7,034	\$109,185
Concrete admixture.....	866	13,703
Filler.....	1,907	41,163
Foundry facing.....	2,426	37,750
Miscellaneous.....	2,515	46,718
Total accounted for.....	14,748	\$248,519
Use not specified.....	6,130	\$101,864
Grand total.....	20,878	\$350,383

^a U. S. Bur. Mines Minerals Yearbook for 1932-33.

The deposits that were examined on the east side of Bear Creek in this district are in the Warsaw. They are not of high grade, being mixed with too much coarse chert.

PRODUCTION AND USES

The tripoli industry is small, the tonnage produced annually ranging between 10,000 and 40,000 tons over the last 10-year period. Table 3 illustrates typical production and value. These figures are for the total market. Individual states producing tripoli during the years listed were

Oklahoma, Missouri, Pennsylvania, Arkansas, Tennessee, Georgia and Illinois, with Illinois producing more than any other state each year.

Table 4 illustrates the chief uses to which tripoli was put during the year 1933. Silica in a fine state is used in a variety of ways but chiefly as an abrasive or as a constituent of an abrasive material, such as scouring powders and soaps. In this same general usage it is employed in polishing and buffing, particularly on metal surfaces. Tripoli, being known to the trade as "soft" silica, is especially adapted to filling wood, rubber and various artificial fabrications. It is widely used as a suspensory in paints, much of the fine Illinois material being used in this manner. Small quantities are used also in glassmaking, ceramics, insulation for small furnaces, for making



FIG. 8.—EXPOSURE OF PURE WHITE TRIPOLI IN WARSAW FORMATION NEAR STOUT, TENNESSEE.

heat-resisting enamel for stoves, and so forth.

Tripoli from the western Tennessee Valley district is well adapted to any of the processes or products that employ soft silica.

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Concentration of Georgia Kyanite Ore

BY R. G. O'MEARA* and B. W. GANDRUD,† MEMBERS A.I.M.E.

(New York Meeting, February, 1936)

IN 1934 the Southern Experiment Station of the U.S. Bureau of Mines in cooperation with the University of Alabama began an investigation of the availability of kyanite ores as a source of raw material for the manufacture of refractories. Under this program, gravity concentration and flotation tests have been made on two varieties of Georgia kyanite ore, mica schist and massive. Most of the attention has been directed to the mica schist ores, as they are more widely distributed and seem to offer greater economic possibilities than the massive ores. It is evident that the preparation of kyanite for use as a refractory material involves other considerations than the mere elimination of impurities. A concentration process that eliminates impurities satisfactorily may have to be followed by some additional treatment, since the final product must be of a coarseness and size ratio that will make it suitable for refractories manufacture. So far in the investigation at the Southern Experiment Station of the U.S. Bureau of Mines only the concentration problem has been considered, and this paper is intended as an interim report.

ORES

The schist ores contain 1 to 9 per cent of kyanite crystals embedded in mica and quartz, with smaller amounts of graphite, garnet and iron oxide and minor amounts of other heavy minerals. The garnet content varies from a trace to a few per cent. The minerals are easily released by moderate grinding. The size of the kyanite ranges from fine needles to crystals $1\frac{1}{2}$ in. long, $\frac{1}{2}$ in. wide and $\frac{1}{4}$ in. thick. The garnet, quartz and iron oxide grains have a smaller mean mesh than most of the kyanite. The maximum size of the garnet crystals corresponds to about 14 mesh and that of the sand grains is considerably less. Most of the iron oxide is still finer, for the minus 100-mesh size is greatly enriched in Fe_2O_3 .

The massive ore consists of radiating masses of kyanite interlocked with quartz, mica, corundum and small amounts of other heavy minerals.

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The sample on which tests were made was about 60 per cent kyanite, nevertheless almost as fine grinding was required for liberation of impurities from this ore as from the schist ores. In the grind, much of the kyanite formed needles or flaky splinters instead of the blocky grains of crystals released from the schist ore.

DEGREE OF GRINDING

The gravity concentration tests on both ores indicate that a high recovery of a low-grade or a low recovery of a high-grade kyanite concentrate can be made. The recovery and grade are closely related. If the ore is ground fine the kyanite is clean, but the recovery is low because needles of the mineral are lost in the middlings. With less grinding the recovery is high but the grade of concentrate is low because of adherent impurities.

Of the two ores the mica schist appears to be slightly more amenable to concentration, for the kyanite does not flake so readily. In the schist ore fairly clean blocky kyanite is liberated by moderate grinding. The individual grains of gangue minerals are released without badly overgrinding the kyanite.

Hydraulic classification segregates the kyanite partly because of the difference of specific gravity and partly because of the difference in mean mesh between kyanite grains and grains of gangue minerals. The coarse spigot products are high in kyanite and the successive spigots are impoverished in kyanite and enriched in mica, sand, garnet and iron oxide. A large part of the iron and mica can be rejected by classification alone. Mica can be rejected effectively from all the spigot products by table concentration. Sand can be separated from the coarse spigot products by tabling, but from the fine products in which the needles of kyanite are associated with the sand, the separation is not so marked, since the shape of the grains defeats the advantage of specific gravity. Garnet, pyrite, and other heavy minerals are segregated mainly in the finer sizes and can be separated in part by tabling.

REPRESENTATIVE TESTS

The following is a summary of a representative classification and table concentration test on a mica schist ore containing 9 per cent kyanite. The ore was ground through 8 mesh in a rod mill and classified into eight spigot products and an overflow. The coarsest product was 64 per cent kyanite; the overflow contained only a trace of kyanite, but 12 per cent Fe_2O_3 . The intermediate products ranged between these extremes. The coarsest spigot product contained 80 per cent of the total kyanite, and the first six spigots, 93 per cent. Each spigot product was tabled and narrow zonal cuts were taken around the table. Of these cuts, from 99.1 to 0.1 per cent sank in a solution of 2.94 sp. gr. By means of

hydraulic classification and tabling; 50 per cent of the kyanite was recovered in a concentrate assaying 85 per cent kyanite. By reclassification and tabling of the middlings, and regrinding and re-treating the middling from this operation an over-all recovery of 67 per cent of the kyanite was made in a concentrate assaying 90 per cent kyanite. A relatively clean mica tailing was separated and a sand-kyanite middling was recovered for re-treatment.

A similar gravity-concentration test was made on the massive kyanite ore. Jig tests were unsuccessful, because of the platy nature of the kyanite and the locking of the coarser sizes. The ore then was ground through 8 mesh, classified and tabled. Zonal cuts were taken around the table. The results were of the same order of magnitude as for the schist ore but the middling losses were slightly higher. The flaky nature of the kyanite made it difficult even to reject a mica tailing free from kyanite.

On both ores the recovery can be increased by stage crushing and re-treating the middlings as long as the particles of kyanite retain a "blocky" shape. When the fine sizes are reached and flakes or needles form, no appreciable separation is obtained. This middling represents a considerable part of the total kyanite in the ore, and varies with the grade of the feed.

Flotation tests have been made on various grades of middlings from the schist ore. All appear to be amenable to beneficiation. In one test on a middling assaying 55 per cent kyanite, over 80 per cent of the kyanite was recovered in a concentrate that assayed 98 per cent kyanite and 1.1 per cent Fe_2O_3 . The graphite was floated first and then the kyanite. The reagents and the amounts per ton of feed used in the rougher were: pine oil, 0.08 lb. per ton; sodium oleate, 0.4 and oleic acid, 0.8. One cleaning with 0.2 lb. of sodium silicate per ton of original feed was required.

Similar results have been obtained in flotation of the massive ore. In this case, the separation is simplified by the absence of graphite.

In general, classification and tabling with careful control yields a fair recovery of a 90 per cent kyanite concentrate. For example, from a 9 per cent ore, 67 per cent of the total kyanite was recovered in a concentrate of this grade. To raise the grade to 95 per cent or better and to improve the recovery, flotation is essential.

DISCUSSION

(W. H. Coghill presiding)

W. H. COGHILL,* Rolla, Mo.—This paper could not have been written 10 years ago, for the good reason that chemists cannot be of as much help in the analysis of products as they can in some of the other industrial fields. Heavy liquids had to be developed as a substitute for chemical analysis. The heavy liquid was used for detect-

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ing the different products and determining the percentage of kyanite, and we did not have heavy liquids in our laboratories 10 years ago. Geologists had used them for many years, but they did not spread the good news or teach us how to use them in working quantities. Their application is a part of the general survey that the Bureau of Mines has been making.

The other day I enumerated the minerals in the industrial group that have yielded more or less to concentration. In some cases the laboratory work has been commercialized; in others commercialization seems to be waiting for the demand. I had, before I got through, a list of 28 nonmetallic minerals that mineral-dressing men had examined and on which they had really made headway in their concentration. Most of the work has been by flotation. Years ago, we thought flotation meant dealing with a few sulphides—iron, lead, copper and zinc. Now flotation has reached out into this nonmetallic field and offers a potential list of 28 minerals.

R. W. SMITH, * Atlanta, Ga. (written discussion).—The problem of the utilization of kyanite from the Georgia kyanite schists demands further research on both the beneficiation of the kyanite and its use in refractory and ceramic bodies. The work described in this paper indicates that kyanite of excellent quality can be prepared from these kyanite schists, but further work is necessary to determine whether or not this can be done at a price that will enable it to compete with other sources of kyanite. Particular attention should be paid to possible by-products, such as flake mica for the artificial roofing trade and garnet for abrasive use, which may lower the beneficiation costs.

Research is necessary on the physical properties of this kyanite and their effect on its use in ceramic bodies. It is to be hoped that the Bureau of Mines will lead the way rather than leave the problems for the industry to work out. The refractory industries are interested in the possibilities of kyanite, as indicated by numerous inquiries for the information available on the extent of the deposits. One Southern refractories manufacturer has experimented on the use of Georgia kyanite and has successfully developed a high-alumina refractory body using about 50 per cent Georgia kyanite. The kyanite for this use does not need to be particularly low in iron or free silica and therefore would require less beneficiation. There is a possibility that in this use the expansion of the kyanite at its conversion to mullite may be useful in counteracting the shrinkage of the clay bond.

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The Kyanite Industry of Georgia

BY RICHARD W. SMITH,* MEMBER A.I.M.E.

(Pennsylvania State College Meeting, September, 1936)

KYANITE, long known to occur in Georgia, did not excite commercial interest until about 1930. Investigation¹ revealed two main types of deposits: (1) separate kyanite crystals embedded in mica schist; and (2) crystal aggregates of kyanite associated with quartz in small and very irregular veins. The second kind at first sight appeared to be the best source of supply. Large boulders or dornicks of kyanite with more or less quartz were found on the surface at many places throughout the Piedmont and Highland sections of the state, but prospecting failed to disclose a sizable vein. These dornicks seem to be a surface accumulation from small veins or lens-shaped masses not more than one or two feet thick, and with a horizontal extent of not more than a few feet. The larger surface accumulation of kyanite dornicks, where not too badly stained by iron oxide, may prove to be marketable, but it is extremely doubtful whether a vein large enough to mine will ever be found.

In 1932 deposits of the kyanite-mica schist were known to occur in two places northwest of Clarkesville, in Habersham County. In 1934, at the request of and in cooperation with the Georgia Geological Survey, the United States Geological Survey made a thorough investigation² of the deposits and found that the two known areas of kyanite-mica schist in Habersham County were but a part of an unsymmetrical, U-shaped belt (Fig. 1) with a western limb extending into Rabun County and somewhat longer than the eastern limb. The total length of the belt is about 30 miles. The width of outcrop ranges from 100 ft. to $\frac{1}{4}$ mile or more. Deposits of kyanite-mica schist of very doubtful commercial importance were found in several other counties. Only the deposits of this kyanite-mica schist belt in Habersham and Rabun counties will be discussed in this paper.

GEOLOGY

The kyanite-mica schist of Habersham and Rabun counties occurs near the northern edge of the Piedmont Upland in metamorphic rocks of pre-Cambrian age³. These rocks probably were deposited as sedimentary

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¹ References are at the end of the paper.

rocks in pre-Cambrian time, but during the periods of mountain building at the end of the pre-Cambrian and again at the end of the Paleozoic they have been intricately folded, intruded with igneous rocks, and intensely



FIG. 1.—KYANITE-MICA SCHIST BELT IN HABERSHAM AND RABUN COUNTIES, GEORGIA.
(After Prindle².)

metamorphosed. A large oval body of quartzite, known as the Tallulah Falls quartzite, occupies the center of a dome-shaped anticline, around which bend beds of gneiss and schist, at places intensely crumpled. These

beds are a part of the Carolina series⁴. In these occur the belt of quartz-muscovite-kyanite schist described by Prindle (p. 10²) as follows:

The principal kyanite deposits of the area are the relatively low grade kyanite mica schists that occupy the belt just described. No sharp boundary can be distinguished between the kyanite schists and the adjoining rocks. The largest crystals occur in the middle part of the belt. In places there are subordinate beds of schist with kyanite separated from the main body by micaceous quartzites. . . . The thickness of the kyanite-bearing schists, so far as observed, ranges from 30 to 60 ft. The crystals are not uniformly distributed through the entire thickness. Beds a few inches thick and crowded with crystals alternate with beds of equal thickness where the crystals are more sparsely distributed, but in general the proportion of kyanite appears to be fairly persistent throughout.

The kyanite schist along the western portion of the belt is underlain by gneisses, which are more resistant to erosion and form the crest of the ridges. East of the gneisses are schists crowded with small garnets and a negligible amount of small kyanite crystals. The amount of kyanite crystals in the main schist belt ranges from 1 to 15 per cent, with a general average of 6 to 8 per cent at places favorable for mining. The flat crystals average 1 in. in length, $\frac{1}{2}$ in. in breadth, and $\frac{1}{4}$ in. in thickness.

The outcrop of the kyanite schist is marked by a surface accumulation of kyanite crystals left behind as the mica and sand were washed away. At places erosion has transported the kyanite crystals into the valleys of the present and former streams, where they form placer deposits of some commercial importance.

ORIGIN

Kyanite has generally been regarded as a product of the regional metamorphism of beds high in alumina⁵. Jonas⁶ favors this origin for the Virginia kyanite deposits, but on the other hand Stuckey⁷ considers the North Carolina kyanite deposits to be due entirely to additional alumina brought in by hydrothermal alteration.

Prindle (p. 37²) describes the origin of the kyanite-mica schists of Georgia as follows:

When rocks are folded, overturned, and thrust one upon another the conditions of temperature and pressure undergo great changes, and minerals become transformed to others in closer equilibrium with the new conditions. Orogenic stress has been emphasized by Niggli as an activating cause to bring about such transformations. Kyanite as a mineral resulting from regional metamorphism could thus be developed.

The increased pressures and temperatures due to the influence of intruding masses of igneous rocks also bring about changes in mineral composition partly by transforming material already present and partly by introducing new material. The influence of hydrothermal metamorphism has been far-reaching and regional metamorphism and hydrothermal metamorphism grade into each other.

Several features of this belt of kyanite schist seem to the writer conclusive evidence that the kyanite crystals owe their origin solely to

regional metamorphism of sedimentary beds rich in alumina. The belt is a stratigraphic unit, and its mineralogy and texture, aside from the size and amount of the kyanite crystals, is typical of the metamorphic rocks that prevail in the area. The proportion of kyanite is fairly uniform throughout the belt and shows no more variation than could be expected of the alumina content of the original sedimentary beds. The only indication of hydrothermal alteration is the scattered dornicks of massive kyanite that are found at a few places in the belt, and the kyanite of these dornicks was probably derived from the surrounding rocks rather than brought in by the hot solutions.

MINING AND MILLING

Only one company, the Georgia-Carolina Minerals Corporation, of Franklin, N. C., has mined and shipped any considerable quantity of kyanite from Georgia. This company in 1932 started to recover kyanite crystals from a surface accumulation derived from kyanite-mica schist at Alec Mountain, 6 miles northwest of Clarksville, Habersham County. The operation is described by Prindle (p. 15²) as follows:

The top soil containing the loose kyanite is shoveled into a flume that carries the material to a crude mill, where the larger crystals are screened out. These are put through a muller in which rubber-covered solid wooden wheels travel over the crystals in a tub and rub off the adhering mica flakes and quartz grains. The crystals are then hand picked to remove fragments of quartz and decomposed mica schist. The kyanite shipped ran about 3 per cent iron [oxide] and 3 per cent free silica.

This operation ceased in 1935, when the residual accumulation was exhausted.

In the spring of 1934 the Georgia-Carolina Minerals Corporation discovered and began to mine a kyanite placer deposit in the valley of a small stream one mile north of Clarkesville. The deposit consists of 1 to 3 ft. of kyanite-bearing gravel overlying decomposed bedrock and overlain by 2 to 3 ft. of overburden. The gravel, composed mostly of water-worn quartz pebbles and boulders, contains in places as much as 40 lb. of kyanite per cubic foot. The southernmost part of the belt of kyanite schist, from which the kyanite crystals evidently have been derived, lies on the ridge just beyond the head of the stream. The natural transportation of the kyanite crystals has freed them of adhering mica and quartz, and the kyanite shipped from this deposit has averaged only about one per cent each of iron oxide and free silica.

The overburden is stripped and the gravel is mined by pick and shovel and wheeled to sluice boxes, which carry it to the washing plant. There the cobbles are screened out and the fine gravel and kyanite crystals are scraped back and forth with hoes in the water running through wide sluice boxes until the clay, sand and fine gravel are washed away from the larger kyanite crystals (Figs. 2 and 3). The smaller kyanite crystals escaping

with the fine gravel are recovered in hand-operated jigs. These consist of wooden boxes containing water in which smaller boxes with screen

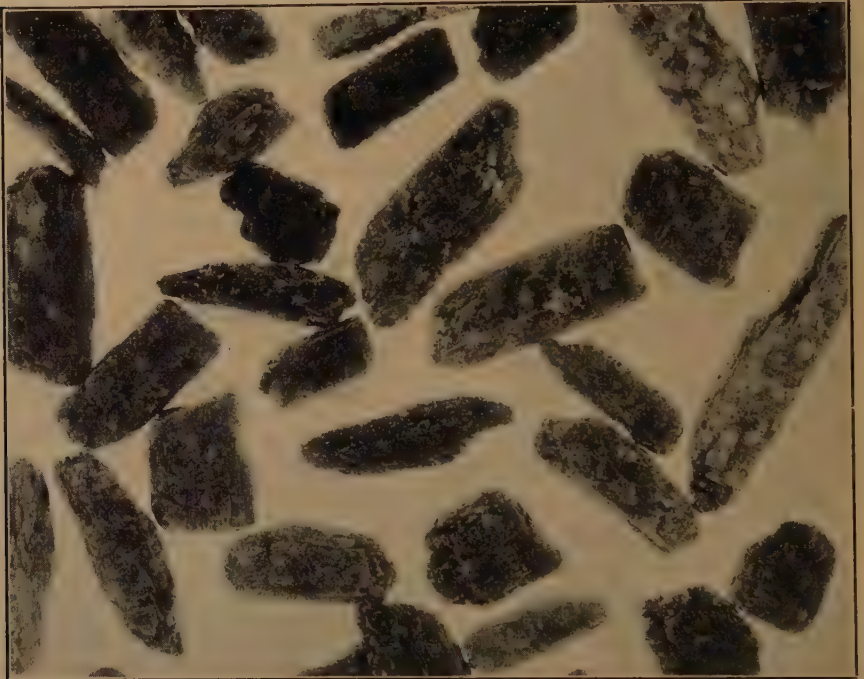


FIG. 2.—KYANITE CRYSTALS FROM PLACER DEPOSIT NEAR CLARKESVILLE, HABERSHAM COUNTY, GEORGIA. NATURAL SIZE.
Largest size shipped. Photo by U. S. Geological Survey.



FIG. 3.—KYANITE PLACER MINE OF THE GEORGIA-CAROLINA MINERALS CORPORATION NEAR CLARKESVILLE.

bottoms, in which the mixture of kyanite and gravel is placed, are moved up and down through the water by the operator (Fig. 4). The greater

specific gravity of the kyanite causes it to settle to the bottom and the quartz gravel is scraped off the top after each operation. The capacity



FIG. 4.—HAND-OPERATED JIG RECOVERING KYANITE AT PLACER MINE OF THE GEORGIA-CAROLINA MINERALS CORPORATION NEAR CLARKESVILLE.

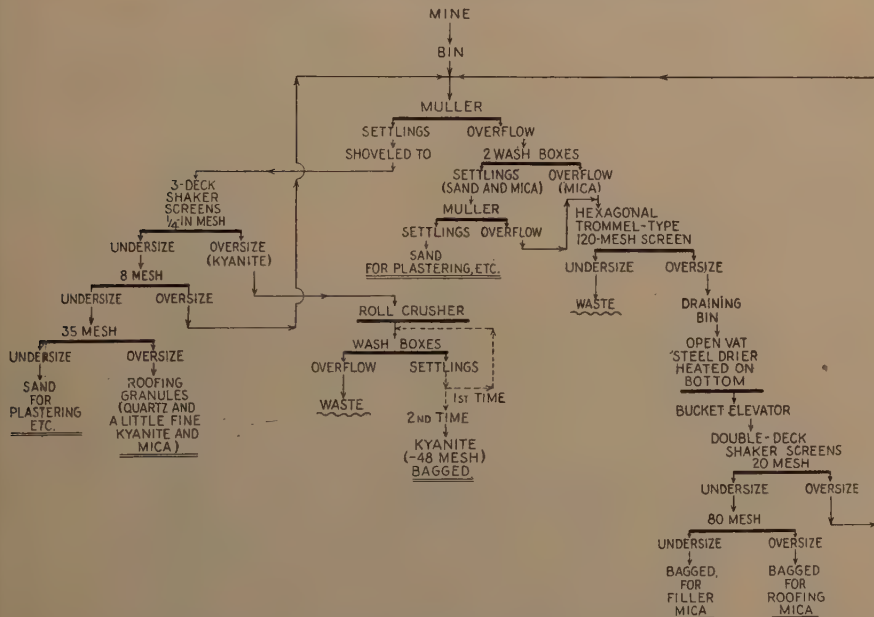


FIG. 5.—FLOW SHEET OF GEORGIA-CAROLINA MINERALS CORPORATION'S MILL ON SUTTON MILL CREEK.

For recovering kyanite, water-ground mica, and other products from a kyanite-mica schist.

of this operation, which employs about 20 men, is $1\frac{1}{2}$ tons of kyanite per day.

From 1932 through 1935, a total of approximately 1100 tons of kyanite was shipped from these two operations.

In 1936, the Georgia-Carolina Minerals Corporation, in addition to carrying on the kyanite placer operation described above, started the recovery of kyanite and other products from partially weathered kyanite-mica schist at two places in Habersham County: one near the southern

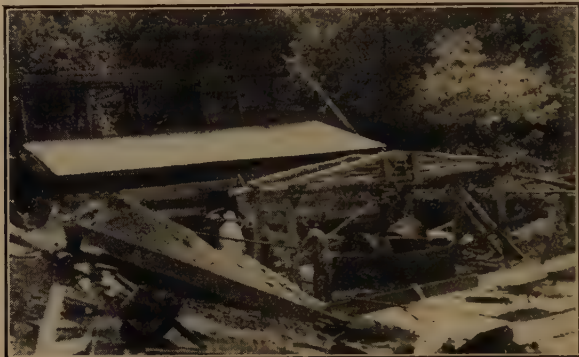


FIG. 6.—PART OF KYANITE-MICA SCHIST MILL OF THE GEORGIA-CAROLINA MINERALS CORPORATION ON SUTTON MILL CREEK.
Water wheel in background and ore bin and muller in foreground.

end of the belt on Sutton Mill Creek, $2\frac{1}{2}$ miles northwest of Clarkesville, and the other at the scene of its earlier activities at Alec Mountain, 6 miles northwest of Clarkesville.

The plant on Sutton Mill Creek is run by water power and has a capacity of approximately $1\frac{1}{2}$ tons of kyanite per 10-hr. day. The flow sheet of the plant is given in Fig. 5, and Figs. 6, 7 and 8 show photographs of part of the plant. Nearly all the machinery has been built locally.

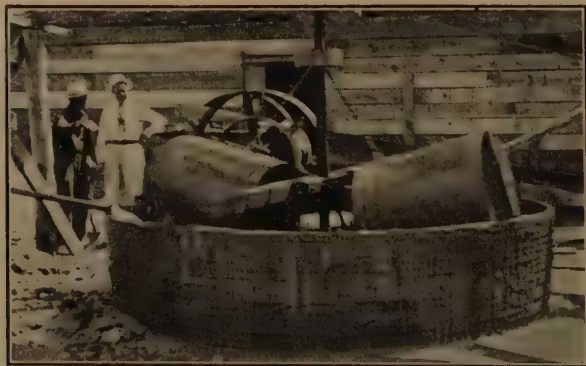


FIG. 7.—MULLER FOR DISINTEGRATING KYANITE-MICA SCHIST, GEORGIA-CAROLINA MINERALS CORPORATION'S MILL ON SUTTON MILL CREEK.

The disintegration of the schist and the separation of the kyanite, mica, and sand are largely accomplished in mullers, consisting of two solid wooden wheels 30 in. in diameter and 48 in. long that travel around a wooden tub 10 ft. in diameter with center overflow. The action of these

wheels traveling over the schist under water flakes off the mica and cleans the kyanite crystals. Further washing and screening results in the production of kyanite crystals, water-ground mica, roofing granules (composed of quartz particles and a little fine kyanite and mica), and two sizes of sand. The kyanite is crushed to minus 48 mesh and bagged for shipment. The mica is dried and screened to two sizes, a fine size for use as a filler and a coarser size for use as roofing mica. Its quality is superior to the dry-ground mica commonly sold for this purpose. No attempt has been made to recover the small amount of amorphous graphite present in the schist.



FIG. 8.—REVOLVING HEXAGONAL SCREEN FOR RECOVERING MICA FROM A KYANITE-MICA SCHIST, GEORGIA-CAROLINA MINERALS CORPORATION'S MILL ON SUTTON MILL CREEK.

The plant at Alec Mountain, because of lack of water power, makes only a partial recovery of kyanite and mica. The schist is treated in two mullers similar to the ones at the Sutton Mill Creek plant. The overflow from the mullers, consisting of mica and fine sand, is treated in wash boxes. The overflow from the boxes is trucked to the Sutton Mill Creek plant for the recovery of the mica. A middling is returned to the mullers, and the tailings, consisting of sand, are discarded. The settlings in the mullers are screened. The oversize from the screen, consisting of kyanite crystals and some undecomposed schist particles, is trucked to the Sutton Mill Creek plant for further treatment. The undersize from the screen, consisting of quartz particles with a little mica and fine kyanite, is discarded.

A plant similar to the one at Sutton Mill Creek, but with a daily capacity of 10 tons of kyanite, will soon be built at a water-power site on the northern end of the kyanite schist belt in Rabun County.

FUTURE OF THE INDUSTRY

The principal market at present for Georgia kyanite, as far as the writer can learn, is for refractories used in the manufacture of glass.

The price asked for this quality of kyanite (one per cent each or less of iron oxide and free silica) has been in the neighborhood of \$30 per ton f.o.b. mines. Philip S. Hoyt, president of the Georgia-Carolina Minerals Corporation, estimates (personal communication) that his low mining and milling costs, together with the simultaneous production of even larger quantities of ground mica, roofing granules, and other "byproducts," will enable him to materially lower the selling price of his kyanite.

The refractories industry has recently been doing considerable research work on the use of kyanite in the production of high-alumina refractories for general use. One southern refractories manufacturer is reported⁸ to have produced experimentally a high-alumina refractory using 50 per cent kyanite in the body. The high price of kyanite alone has prevented him from going into production. Kyanite, because of its expansion at the point of conversion into mullite, is especially suitable for incorporation into plastic refractory ramming mixes. A material drop in price of Georgia kyanite would undoubtedly result in a greatly increased market.

Preliminary reports^{9,10} on experiments on Georgia kyanite being made by the U. S. Bureau of Mines indicate that nearly iron-free kyanite, suitable for use in white ware, glass, and other high-quality ceramic products, can be recovered from the Georgia kyanite mica schists by flotation and other standard ore-dressing methods. The cost of recovery of kyanite of this grade from the Georgia deposits will probably be considerably higher than that of the operations described above.

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DISCUSSION

V. L. MATTSON,* Burnsville, N. C. (written discussion).—In commenting on the extreme purity of the placer concentrates made from the Habersham deposits, I

* Celo Mines.

should like to call attention to the importance of the grain size of the samples tested. I have examined a number of selected crystals from both the Georgia and Carolina kyanite schists and have observed that it is rare to find a kyanite crystal free from quartz and mica inclusions. This is particularly true of the larger crystals. Recently we tested a number of kyanite crystals from Habersham County in our laboratory. The crystals were cleaned to remove any adhering impurities. These crystals would readily sink in a methylene iodide solution having a specific gravity of 3.4. The crystals were then ground to pass a 48-mesh screen, and approximately 10 per cent of the ground material floated in the same solution. Microscopic examination showed the float material to be principally quartz with a little muscovite. The specific gravity of the unground crystals was about 3.47, which would indicate a purity of about 90 per cent kyanite. These observations have led me to conclude that in most cases it is necessary to grind the kyanite crystals from the schist formations in order to secure complete unlocking of included quartz particles.

J. L. STUCKEY,* Raleigh, N. C. (written discussion).—With the growing evidence of hydrothermal kyanite, it is hardly probable that Mr. Smith's conclusion that the kyanite crystals in the schist owe their origin solely to regional metamorphism will be universally accepted. He quotes three references to substantiate his ideas and in each there is material that contradicts them. Clark⁵ states that kyanite is often embedded in quartz and has been reported in limestones. Jonas⁶ states that the kyanite in the pegmatite dikes of central and southwestern Virginia is of igneous origin. In the report by Prindle² reference to hydrothermal kyanite is found on pages 19, 22, 31 and 37. Bayley¹¹ has also thrown doubt on the regional metamorphism idea for the origin of kyanite in the Tate Quadrangle.

A number of minerals formerly classed as due to regional metamorphism have been definitely removed from that classification in recent years. While kyanite may not have been completely removed from that classification, considerable doubt has been raised as to its mode of origin. A considerable amount of laboratory work, both chemical and microscopic, carried on in connection with carefully done field work seems advisable in solving the problem. A good start has been made in Georgia with a splendid piece of field work, and it is hoped that the laboratory work may be done and all the facts made available.

G. R. MANSFIELD,† Washington, D. C.—I have visited the Georgia deposits. The kyanite appears to be very abundant, but the surface accumulations a foot or two thick are much richer than the solid formations. While the demand is small the product can be recovered easily from the surface deposits, but if a larger demand develops and these are exhausted, the solid rock must be mined, and this will involve more elaborate milling methods. Will Dr. Vaughan please describe the ceramic applications of kyanite?

W. H. VAUGHAN,‡ Atlanta, Ga.—We have to consider both the physical and chemical properties. At 1400° C. kyanite increases in volume. It is found advantageous to use it in making rammed refractories. When this type of refractory is used, there must be no movement between it and the wall on which it is placed. When kyanite is added to fire clay the expansion of the kyanite compensates for the shrinkage of the clay and a change in volume occurs, of not more than 1 per cent up to 2700° F.

* Professor of Geology, North Carolina State College of Agriculture and Engineering.

¹¹ W. S. Bayley: *Geology of the Tate Quadrangle*. Georgia Geol. Survey *Bull.* 43 (1928) 70-73.

† Geologist, U. S. Geological Survey.

‡ Department of Ceramic Engineering, Georgia School of Technology.

Thus expansion is not always a disadvantage. In manufacture of hotel china, it works the same way, compensating for the shrinkage of the clay.

E. F. BURCHARD,* Washington, D. C.—I noted on one of the slides a crystal of corundum associated with the kyanite. This shows evidence of high-temperature origin.

R. W. SMITH (written discussion).—Professor Stuckey has first noted but then ignored the fact that I postulated a regional metamorphic origin for only the kyanite crystals of the kyanite-mica schist belt of Habersham and Rabun Counties, Georgia, and in the last sentence of the section on origin I have clearly indicated a hydrothermal origin for the massive kyanite found in that belt and, by inference, for the massive kyanite found at many other places in the crystalline area of north Georgia.

The references quoted by Professor Stuckey are, with one exception, not applicable because they refer in general to kyanite of the massive type. The reference to Jonas concerning kyanite in pegmatite dikes and to Clarke, who speaks of "long bladed crystals imbedded in quartz," are both of this massive type. The reference cited to page 19 of Prindle's report reads: "Massive kyanite was found [in Habersham and Rabun Counties] only in the belt of kyanite schist. Tourmaline is associated with it, and the origin is regarded as hydrothermal," which clearly refers only to the massive kyanite in that belt. Similarly, on page 22, Prindle is referring only to massive kyanite. The citation to page 31 refers to the kyanite at Graves Mountain, in Lincoln County, where kyanite is found in or closely associated with quartz veins cutting a quartzite and is therefore suggesting a hydrothermal origin for that particular deposit. The citation to page 37 refers to Prindle's chapter on the origin of kyanite, in which he says: "The evidence available indicates a hydrothermal origin for the massive kyanite . . . It is believed further that to this action has been largely due the *migration of kyanite material in the schists to form the massive kyanite in the veins.*" (The italics are mine.) This in no way contradicts my statement that "the only indication of hydrothermal alteration is the scattered dornicks of massive kyanite that are found at a few places in the belt, and the kyanite of these dornicks was probably derived from the surrounding rocks rather than brought in by the hot solutions."

The citation to Bailey refers to a description of massive kyanite and quartz veins bordered by massive kyanite occurring in the Tate quadrangle in a mica-kyanite schist much more siliceous and lean in kyanite than the kyanite-mica schist belt of Habersham and Rabun Counties. Bailey attributes both the massive kyanite and the kyanite crystals in the schist to hydrothermal origin, the silica coming up from depth with the heated waters and absorbing alumina from the schists. Is it not possible that here too, although the evidence is not as conclusive, the kyanite of the schists resulted from regional metamorphism and the massive kyanite bordering the quartz veins from hydrothermal alteration with transportation and redeposition of the alumina from the schists to form the massive kyanite?

Surely Professor Stuckey does not mean to imply that all kyanite must have the same origin, and that if the origin is hydrothermal in some cases or for some types, it must be hydrothermal for all cases or types. Further field and laboratory work in detail not practical for a State geologic survey might disclose more evidence, as he has pointed out, and I should very much like to see him undertake the problem. But meanwhile he has not answered the arguments I advanced for a regional metamorphic origin for the kyanite crystals in this particular belt of kyanite-mica schist.

* Mining Geologist, U. S. Geological Survey.

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